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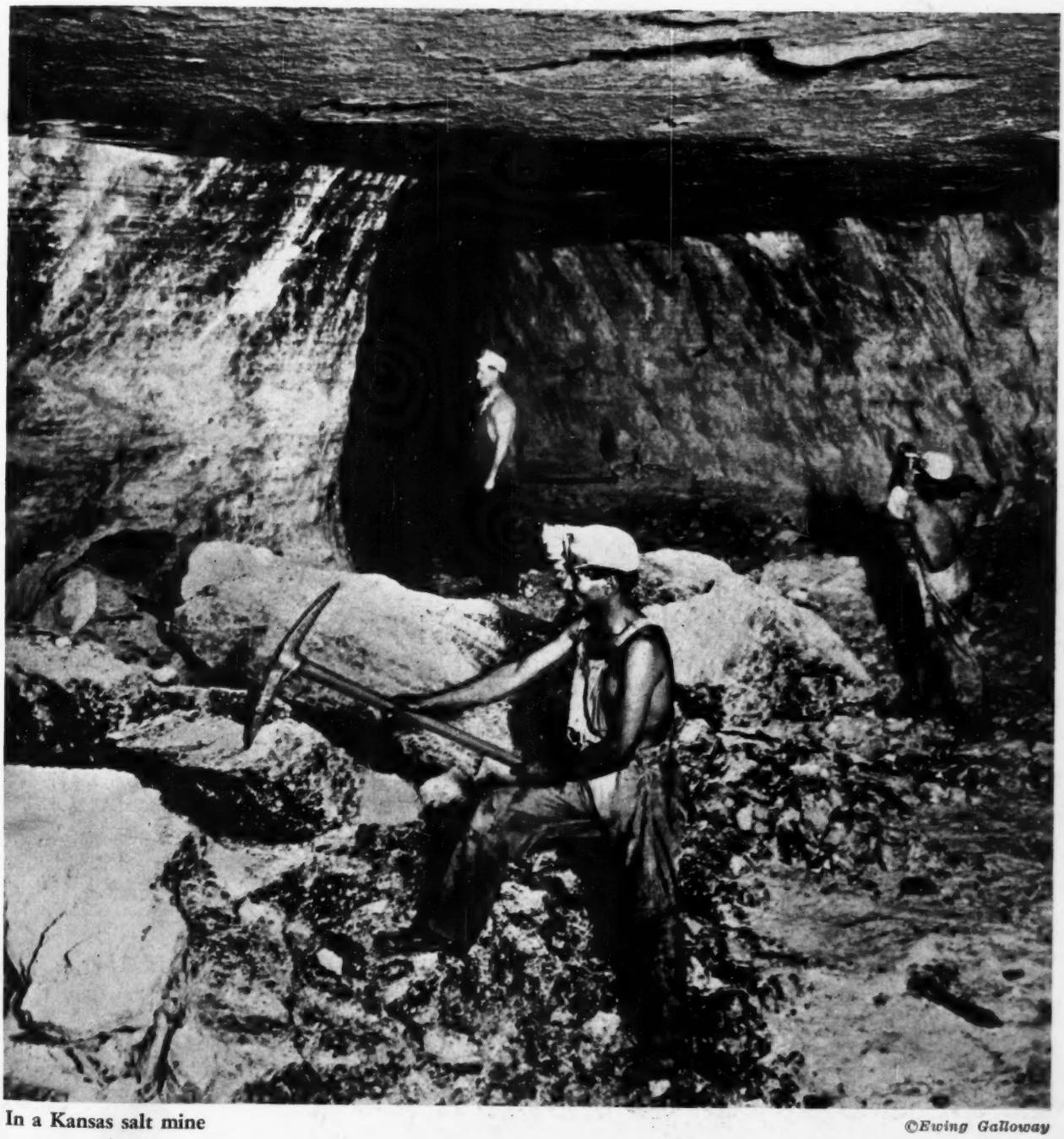

In a Kansas salt mine

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April 13, 1931

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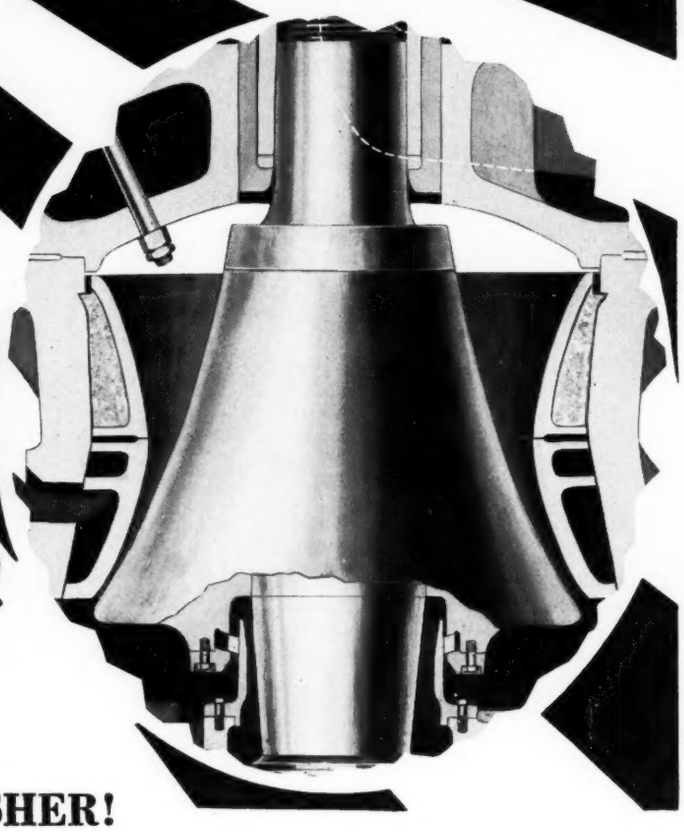

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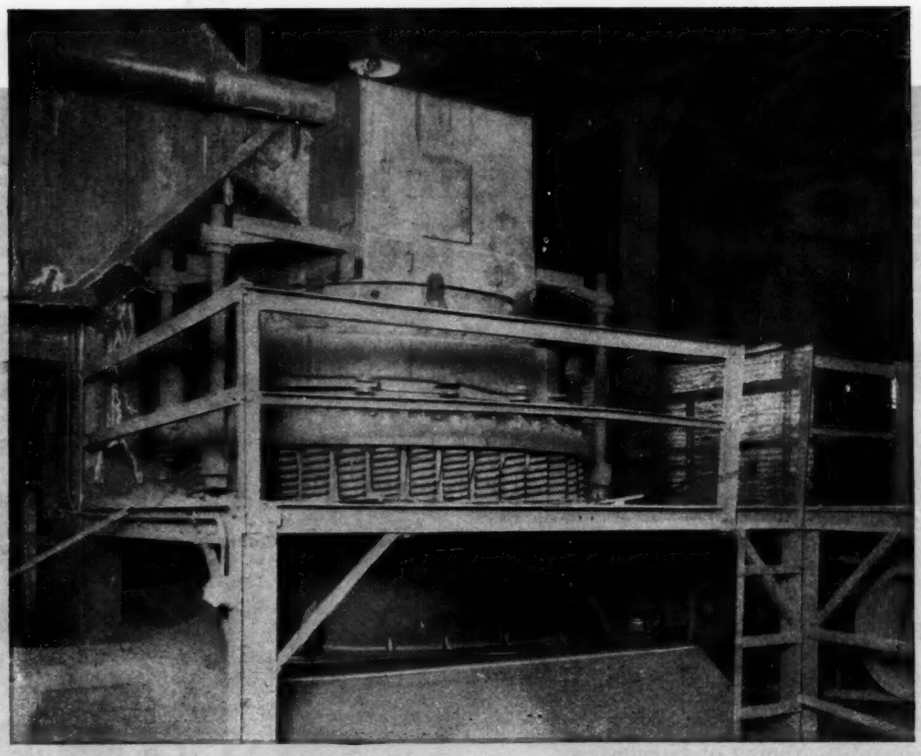

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Engineering and Mining Journal

April 13, 1931 A. W. ALLEN, *Editor* Vol. 131—No. 7

Industrial and Personal Forecasting and Planning

IN TAKING STOCK of current economic conditions one is reminded of the lack of intelligent foresight that characterized the preceding period of prosperity, a neglect that failed to take advantage of even the most elementary principles of insurance. Governmental, financial, economic, and academic experts formed a consensus of unhealthy optimism that was based on conditions then prevailing, since proved to be economically unstable; with the result that the average citizen was given the impression that his thinking had been done for him, and that his major obligation in return for a high standard of living was to mortgage the future to the financial limit of his credit.

Many who have studied the suggestions for saner business conduct to which attention has been called in former issues of *Engineering and Mining Journal* will readily admit the importance of the part that careful planning will play in bringing about a readjustment of economic affairs. To those who are interested in following this phase of the subject a little deeper may be recommended for study and inspiration an outline of the principles governing successful industrial forecasting, as enumerated by Mr. S. L. Andrew at the dedication of the James Ward Packard Laboratory at Lehigh University last fall and republished in the *Bell Telephone Quarterly* for January, 1931.

Forecasting is defined as having for its objective the anticipation of those changes that are characteristic of the modern economic system, "to appraise their probable importance, and to translate them into estimates and plans and policies"—for future guidance. Mr. Andrew differentiates between short-term forecasting and long-term forecasting, the former being concerned primarily with seasonal fluctuations, the latter being needed to reach sound conclusions in regard to the rational status of industry, and the rate of growth or business volume consistent with that status. The achievement of successful forecasting involves an approach from two distinct angles—the statistical and the philosophical. Due recognition of intangibles is needed, by one with an "insatiable curiosity" as to the reasons for change, with an "innate flair for the detection of those undercurrents in economic behavior not visible on the statistical surface of the stream. . . ." In both classes of forecasting is seen the need for "statistical analysis and measurement, on the one hand, and economic interpretation and judgment, on the other."

Forecasting, like budgeting, is as necessary for the individual as it is for business as a unit or as an aggregation of units. It emphasizes, when employed, the need for rational acquisition as well as rational expenditure. It focuses attention on the need for adequate provision to meet conditions indicated—as possible if not always probable—by the exercise of mature judgment, unswayed by political catch phrases and resistant to the hypnotism of half truths. The future can be faced with confidence if adequate foresight is blended with intellectual honesty in facing facts, and supported by the zeal and skill that the people of the United States possess in such abundant measure.

Copper Refining Becomes More of an International Art

ALTHOUGH the first successful plant for the electrolytic refining of copper was built and operated in England in 1869, by the turn of the century the United States had become the leader in that particular field of metallurgy. The reasons for this leadership are immediately apparent. The United States was not only the largest producer of primary metal, but it also constituted the greatest market for use of copper in the electrical industry, which requires copper of exceptional purity. Its technicians, recognizing the advantages of the free interchange of their knowledge, did not hesitate to publish the results of research as well as of operating experience.

Within the last ten years the United States has not maintained its proportion of total world mine output of copper, even though the volume of its production has, on the whole, increased. New properties in other countries have taken an increasingly large share of the copper market. But the United States had, until recently, maintained its position in the refining business. Even that position is now threatened. The American Bureau of Metal Statistics has released figures which show that in 1930 the United States contributed only 68 per cent of the world's electrolytically refined copper, compared with 75 per cent in the previous year. No matter what turn the copper market may take, this percentage will probably continue to decrease. Canada, Chile, Belgium, Australia, and other primary producers are desirous of refining their own output. Northern Rhodesian blister will probably be refined in England when production reaches sizable proportions.

The trend is one that cannot be stopped. Man is a

creature quick to imitate, the world over. Success in any field breeds the desire for duplication. If the United States—through electrolytic copper refineries, automobile factories, moving-picture studios, and countless other productive facilities—has led the way in an improvement of standards of living, it must be prepared to see other countries adopt its technique, even use its own technicians. To some observers, such tendencies indicate competition. A recent editorial in *The New Republic* on the subject of copper suggests that an international struggle for the market is imminent. That is a possible interpretation, but another is the interpretation which *Engineering and Mining Journal* prefers. Even as water seeks its own level, so do the laws of economics. They are hampered and restrained in many places by tariffs and subsidies, but in the end copper will be refined, automobiles will be built, moving pictures will be made, where the combination of source and market is most economic. The combination still holds in the United States. Its reserves of copper are large, its consumption of the metal is the greatest in the world. Those factors are sufficient assurance that the country's refineries will still play a large part in the copper industry.

Increasing Silver Value by Hallmarking

THE United States and other modern countries have been largely responsible for silver's steady loss of dignity throughout the twentieth century as a precious metal. The imprinting of the adjective "sterling" adds nothing of interest or value to the article on which it appears, in the eyes of the average citizen. It carries no government guarantee.

In Great Britain, on the other hand, a silver article that has been marked at one of the government assay offices, at a negligible cost, is prized by its possessor. The hallmark adds greatly to sales appeal. The date of stamping—even back to the initiation of hallmarks in the fifteenth century—can be determined by the type of letter used. A distinguishing mark designates the assay office at which it was stamped; the so-called "sovereign's mark," introduced in 1300—a lion passant—is a guarantee of fineness; and for a while the head of the reigning monarch appeared on all stamped silver articles. Consequently, old silver, if marked, appreciated in value, irrespective of current market price for the metal, and this appreciation adds to the interest shown in all articles made of the metal, even if recently fabricated. Why not an American hallmark for sterling silver, with some esthetic or interest appeal, to restore its dignity as a precious metal?

Those who think only in millions and billions of dollars and ounces will have little patience with the recommendation that the sale of silver as an industrial and ornamental metal be encouraged. On the other hand, while economists argue and politicians dispute the practicability of this or that plan to improve the status of silver, the average citizen may find satisfaction in doing the minor tasks that lie close at hand. And the first and most obvious of these is to correct the anomaly that lower silver prices in 1930 brought smaller sales of silver goods and diminishing profits to all concerned, particularly the silver miner.

Tenacity of Purpose Governs the Progress of Safety

KEEP everlastingly at it! This is the battle cry and the expression of the fondest hope of all sincere proponents of safety, a cry that would carry the records of accident prevention endeavor to new heights, and a hope that the legions of the safety-minded be augmented until every individual is enlisted in the war against accidents. And a war it is; one that is more severe, from the standpoint of human lives sacrificed, than the World War. For example, and considering only one group of accidents, recent statistics reveal that, compared with the 50,510 men lost in combat by the American Expeditionary Forces in eighteen months, 50,900 persons were killed by motor accidents in the United States during a similar period ended December 31, 1930. The situation is alarming; and who has not felt its pall through the accidental death or serious injury of a relative or friend?

Inasmuch as the application of mechanical devices will continue to increase generally, reduction in the number of accidents that accompany their use is dependent on suitable education in safety for the public at large. How this education can be brought about is the problem. The schools can do much with the younger generation, but the major part of the task falls on business and industry. Safety learned in the plant will penetrate into the home, and ultimately will govern personal action elsewhere, as on the highway. But, to be effective, accident prevention education must be continuous and persistent; a desire to prevent accidents must be built up and maintained in the individual.

Numerous companies and industries have established excellent safety records that may be attributed to the "keep-everlastingly-at-it" attitude, the achievement of the Portland Cement Association in 1930 being particularly noteworthy. Beginning with a record of one plant in 1924 that finished the year without a lost-time accident, the record increased to 2 plants in 1925 and 1926, 10 in 1927, 16 in 1928, 26 in 1929, and 43 in 1930. This last record of 43 plants without a single lost-time accident was made with a total of 150 plants operating.

In speaking at a recent meeting of the operators of another industry, Mr. A. J. R. Curtis, assistant to the general manager of the Association, explained that the success of its accident-prevention work was directly to the credit of an efficient committee, headed by the president of one of the member companies, who, while still a boy, lost his father through an accident that occurred while the father was at work. "He," said Mr. Curtis, "knows the hardship and suffering and loss of education that accidents often cause; he is heart and soul in the work." The chief executives of the member companies, who support the committee and interest the men in the work, making them feel that the companies' primary motive behind safety endeavor is humanitarianism, are also largely responsible for the success attained.

As the results indicate, the Portland Cement Association's safety work has amply demonstrated its effectiveness. Administration and operation are intimately based on the "keep-everlastingly-at-it" principle.

Fig. 1—Part of the South pit

Open-Cut Mining of Fireclay

Laboratory Control and Technical Supervision of All Phases of Extraction and Processing Operations Insure Quality and Uniformity of Manufactured Product

H. D. Keiser
Assistant Editor

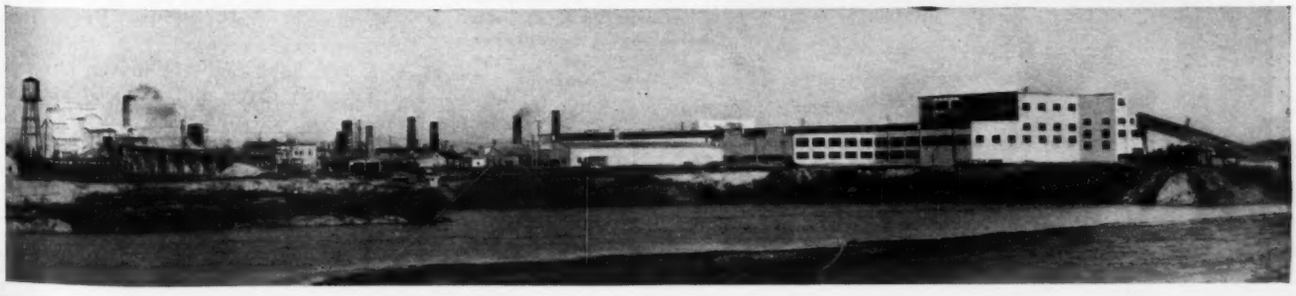
THE world's largest open-cut fireclay-mining operation is probably that at Mexico, Mo., where the A. P. Green Fire Brick Company has a pit 800 ft. wide and 1,700 ft. long. At this point, which is about midway between St. Louis and Kansas City, the company controls a tract of 1,000 acres, and since 1910 it has been manufacturing various refractories there, including (1) diaspore, or high-alumina brick, commonly known as super-refractories; (2) high-grade fireclay refractories; (3) plastic firebrick; (4) high-temperature bonding mortar. Raw material for the super-refractories is obtained from the company's deposits in the central Missouri diaspore field. Plastic and semi-flint clay for the manufacture of the other products are mined at Mexico, and flint clay, also required, is obtained from near-by deposits in Calloway and Montgomery counties.

At the aforementioned pit, known as the South pit, the clay deposit averages 35 ft. thick. It lies on a pyritic quartzite and below 20 ft. of overburden that consists of, first, 14 ft. of soil, then 5 ft. of sand and gravel, and, finally, immediately above the clay, 1 ft. of limestone. The upper layer of clay, averaging 12 ft. in thickness, is plastic clay, and below this is the semi-flint clay, the thickness of which averages 23 ft. Both clays are light gray to nearly white in color. The plastic clay, however, is comparatively soft, containing 6 per cent free moisture and 10 to 11 per cent chemically combined. It has a coarse grain, and pyrometric cone-equivalent tests show it to have a cone-fusion point of Cone 31. The semi-flint clay is harder, coarser, and more refractory, with a cone-fusion point of Cone 33.

Prior to opening the South pit in 1928, its entire area, as well as outlying portions of the clay deposit, was prospected with a 2-in. diamond core drill, made by the E. J. Longyear Company. In this work a uniform drilling pattern was followed.

In current operations at the pit, the overburden is removed with a Marion No. 37 electric shovel, equipped with a 1¾-yd. dipper. The depth of the single cut employed is the depth of the overburden, or 14 ft. Removal of overburden is kept three years in advance of estimated clay requirements. From the shovel, the overburden is carried

Plant of the A. P. Green Fire Brick Company, at Mexico, Mo. The new No. 3 tunnel-kiln unit is seen at the right; in the foreground is an abandoned clay pit filled with water



three-quarters of a mile in 4-yd., wooden, side-dump cars to a dump, where low land is being filled in. Trains of ten cars are drawn by 20-ton Davenport steam locomotives over 36-in.-gage 60-lb. rails. The tracks on the dump are moved laterally, as filling in of the low land proceeds.

Normally, six benches, each 6-ft. deep vertically, are worked simultaneously in the pit, the clay being blasted from the benches to facilitate loading operations. The drill holes are $\frac{3}{4}$ in. in diameter and 7 ft. deep. They are spaced 6 ft. apart, parallel to and 6 ft. from the face of the bench, the resulting drilling pattern being a series of 6-ft. squares. With the "Little Giant" electric auger drills, made by the Chicago Pneumatic Tool Company, that are employed, a 7-ft. hole is completed in ten minutes. The holes are charged with 35 per cent gelatine ammonia dynamite and fired with cap and fuse. Large blocks of clay produced in the course of blasting operations are reduced by mud-capping. Maximum size of the broken clay is 12x12x18 in. Blasting costs average $1\frac{1}{2}$c. per ton of clay mined.

With the exception of particularly high-grade clay that is selected and loaded by hand, the blasted material is loaded into 2-yd. steel Koppel cars by three Northwest electric shovels, equipped with $\frac{3}{4}$-yd. dippers. Trains of eight cars are moved over the 36-in.-gage 60-lb. rail lines in the pit by $3\frac{1}{2}$-ton Whitcomb and 5-ton Brookville gasoline locomotives. On reaching the bottom of the 500-ft. incline leading out of the pit, the locomotive is uncoupled from the train and the cars are pulled up the 12 per cent grade by means of a $\frac{3}{4}$-in. cable from an electric hoist, operated by a 125-hp. G.E. motor that is placed on the ground level at the top of the incline. The operator of the hoist keeps a daily record of the number of cars of clay and of overburden that are removed from the pit, and this record, together with periodic surveys of the pit, provides an accurate estimate of the quantity of clay and overburden excavated.

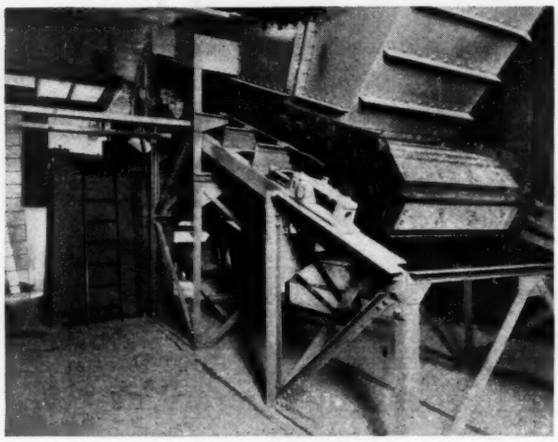


Fig. 2—Apron conveyor that feeds the clay to the crushing rolls

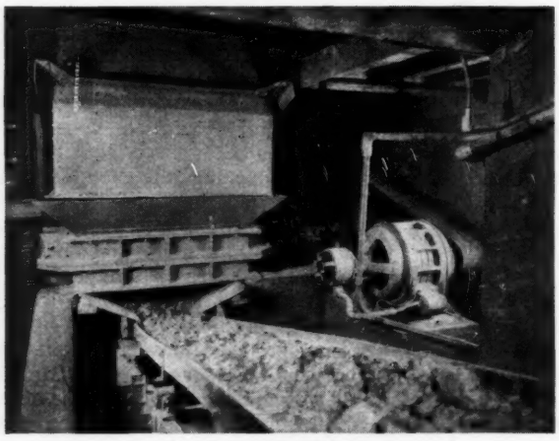


Fig. 3—Crushing rolls and lower end of inclined steel conveyor

Fig. 4—Steel conveyor without load, as seen looking up conveyor gallery from near crusher

The pit is relatively dry, inasmuch as no seepage occurs and no underground watercourses prevail in the strata mined. What water must be removed is limited to natural precipitation, which averages 39.3 in. annually. This water is directed to several sumps, whence it is pumped to waste on the surface. The pumps employed for this purpose are all electrically driven centrifugal units, with 3-in. to 8-in. discharge, and include two 6-in. single-stage Allis-Chalmers pumps, each rated at 1,400 g.p.m. against a 70-ft. head; an Economy single-stage unit, rated at 250 g.p.m. against a 70-ft. head; a standard Lewis unit; and a Fairbanks-Morse.

From the pit, the trains of clay are hauled about a mile by 20-ton Davenport steam locomotives either directly to a 25-ton subsurface bin at the crusher building or to a large storage pile, where the clay is dumped for use during the winter, when severe freezing weather makes pit operations inadvisable. Track on the storage pile is moved laterally as the pile is built up. A Northwest shovel and trains of Koppel cars are employed in moving the clay from the storage pile to the bin at the crusher building.

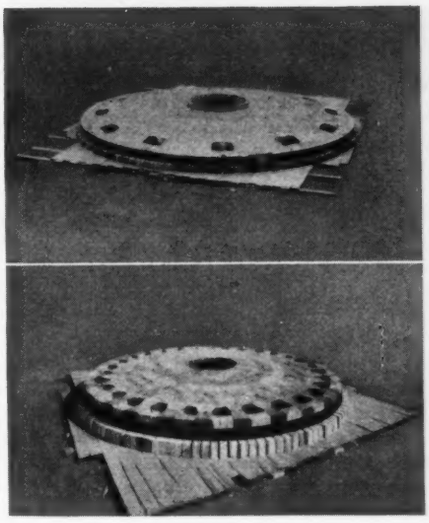
Under normal conditions 1,000 tons of clay is mined daily on a ten-hour shift, with a total of 100 employees in the mining department.

From the bin at the crusher building, a 36-in. Stephens-Adamson steel apron conveyor, shown in Fig. 2, feeds the clay to a set of spaced toothed-type crushing rolls, 24 in. in diameter. These rolls, made by the Eagle Iron Works, have a capacity of about 150 tons per hour and are driven through a Texrope drive and a speed reducer by a direct-connected 75-hp. G.E. synchronous motor. The $2\frac{1}{2}$-in. product of the rolls, as shown in Fig. 3, falls onto a specially processed Swedish steel conveyor, $\frac{5}{16}$ in. thick and 24 in. wide, made by the Sandvik Conveyor Manufacturing Company, that is one of

the most interesting installations at the A. P. Green company's new tunnel-kiln plant, which was placed in operation last June. Speed of travel of the conveyor is 350 ft. per minute, distance between centers 240 ft., slope 15 deg., and capacity 150 to 200 tons per hour. The smooth surface of the conveyor is supported on ball-bearing-equipped wheels that are lubricated by a Zerk alemite system. A 25-hp. G.E. motor drives the conveyor through a speed reducer. Fig. 4 shows the conveyor unloaded, as seen looking up the conveyor gallery from near the crusher.

At the discharge end of the inclined steel conveyor, the clay falls onto a horizontal conveyor, 140 ft. between centers, and with a speed of 300 ft. per minute. Movable plows or scrapers discharge the clay into a row of storage bins, and from these the clay is drawn as required for the manufacture of the various refractories made at the plant, which has a normal daily output of 250,000 9-in. equivalent brick.

A visitor to the plant cannot fail to be impressed by the laboratory control exercised over all operations, from the time the clay deposit is prospected until the manufactured product is shipped.



Roaster hearths

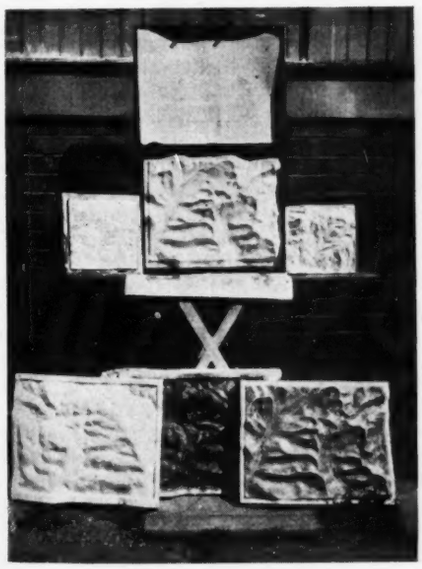
Above—Assembled roaster hearth at the plant of the A. P. Green Fire Brick Company prior to its shipment to an Idaho mining company. This hearth is made up of 5,124 separate brick and tile. Below—Another roaster hearth assembled prior to shipment to a copper smelter in Arizona.

For instance, sampling in the pit precedes mining by about a week. Representative 70-lb. samples are taken every 25 ft. along the face of each bench under the supervision of a ceramic engineer. These samples are sent in canvas bags to the laboratory for assay, each bag carrying a tag that shows the source, date, and apparent grade of the sample. Assay values, as well as the results of various physical tests, are sent to the ceramic engineer, who, through the pit superintendent, directs operations in the pit on the basis of these laboratory determinations. In the manufacturing processes the same degree of control to insure quality and uniformity of product is exercised by the laboratory, which is in charge of a skilled director of research.

Acknowledgment is made to Mr. A. P. Green, president, and Mr. J. Harrison Brown, vice-president and general manager, of the A. P. Green Fire Brick Company, for permission to visit the company's plant and to prepare the foregoing account. For the information presented therein I am indebted to Mr. Roy Creasey, assistant general superintendent.

■ ■ ■

New Method for Making Relief Models

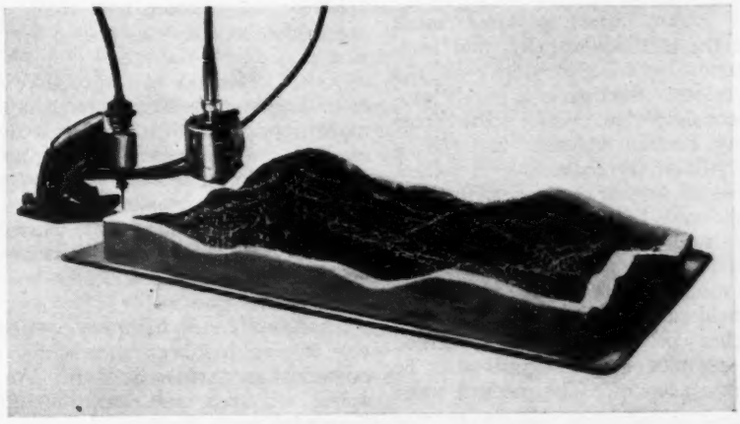


Finished relief models

Relief models can be put together quickly, accurately, and cheaply by the following method devised by B. B. Ellerbeck, inventor and mechanical engineer, of Salt Lake City. A topographic map, enlarged by photographic projection or by photograph, is used as a base for the relief model; the topographic maps of the U. S. Geological Survey are enlarged from five to ten diameters. The enlarged contour map is pasted on a surface of flat boards, and holes of equal depths are drilled along the contour lines at distances varying from one-eighth to three-eighths of an inch. A small high-speed drill, shown in one of the accompanying illustrations, is used for this purpose.

Straight pieces of wire, cut to lengths corresponding to the contour lines, are inserted in the holes along these lines. Plastic material, composed of sand mixed with oil, or of plaster, if permanency is desired, is placed in the spaces between the wires. This filler is then smoothed off even with the top of the wires, giving, in relief, a miniature of the area represented by the original topographic map.

From this master model any number of duplicates may be made. They are cast hollow and are light in weight. The surface may be colored to designate geological formations, wooded areas, lakes, streams, roads, and mines. A coating of Damar varnish gives a lustrous finish. Such a model will show at a glance the relative elevations and similar characteristics of the terrain, position of mines, roads, lakes, and creeks, and other topographic features of importance to the highway, hydraulic, mining, or irrigation engineer.



Model with wires ready for plastic filling material

SHRINKAGE STOPING

Morris J. Elsing

APPROXIMATELY 8,000,000 tons of ore was mined by shrinkage stoping in the United States during 1929, according to the U. S. Bureau of Mines.[1] This tonnage came from 46 mines which use the method for all or a part of their production. These mines are scattered all over the United States and Alaska. The method is extensively used in Mexico, Canada, and other countries.

The shrinkage method of stoping can be applied to orebodies having a wide variety of shapes and sizes. One necessary characteristic for its satisfactory application is that the footwall of the orebody must have a dip steeper than the angle of repose of the broken ore. Walls and orebody itself must also have good standing characteristics, and the ore must not require much sorting. With these main conditions fulfilled, the method can be used satisfactorily, with some minor exceptions. If the ore is such that, after it is broken, it tends to pack and hang up, and will not readily run from chutes, the method cannot be used to best advantage. When the broken ore oxidizes sufficiently to interfere with its recovery in the concentrator, use of the method might be inadvisable.

Irregularities in the boundaries of the orebody are permissible, provided they are steeper than is the angle of repose of the broken ore. Mining horizontal offshoots from the main orebody offers no difficulty, but, when being drawn, complete recovery of the broken ore resting upon such a bench is impossible. Shovelers can, of course, be sent into the stope when the broken ore has been drawn down to a point a little below the level of such a bench, and this ore be shoveled into the main part of the stope. Such procedure is sometimes dangerous, however, and always causes additional expense.

An objection often made against the method—namely, that sorting cannot be done in stopes—is not often serious. Sorting is a purely economic consideration. Waste sorted from ore may contain valuable material, a part of which is recoverable if milled. The cost of sorting in stopes, the cost of additional handling, hoisting, and milling, with a credit for values recovered from the material which might be removed from the ore, are the factors for consideration. In addition to this, the higher cost of stoping by a method which permits sorting must also be taken into account. The modern trend of operations, wherever possible, is to provide larger milling capacity, to work the mine on a larger scale, and with larger-scale operations to work lower-grade ores at reduced costs. On final analysis, sorting and milling are similar operations, the former being done by hand and the latter by machinery. In every line of endeavor machinery is replacing hand labor, with consequent lower cost, and this is often true in hand-sorting versus milling.

Where the waste in the broken ore is actually barren, sorting can be satisfactorily done at the chutes under certain conditions. This is accomplished by having an extra car at the chute. As the ore is being loaded, waste boulders can be picked from it and thrown into the waste car. This is done to best advantage where cars are loaded individually by a hand trammer or chute tender and a train is made up in this way. In many operations the ore is hand-trammed to ore pockets, where it is dumped onto grizzlies, at which point another opportunity to sort out the larger waste boulders presents itself. Such a method has been satisfactorily used, especially where work is done on contract. A higher price is paid per car for waste than for ore, as the waste is obtained only with additional effort. If a price incentive is provided, such sorting can be made really effective.

Under certain conditions where the orebody contains occasional large waste masses or boulders, they can be broken more or less separately and delivered to a chute and thus removed from the stope. In narrower veins low-grade and barren pillars can be left, but unless these are properly shaped they tend to hang up ore in the same way as does a bench.

Because of the relative costs of milling and smelting, sorting is much more important when mining a direct-smelting ore than when mining a milling ore. The cost of smelting is generally so high that the stoping method which permits sorting is usually warranted, because the higher cost of stoping is more than offset by a lesser cost for smelting.

Specific Uses of Shrinkage.—Shrinkage stoping has been successfully used on veins as narrow as 3 ft. An example of such an operation is the Smuggler, at Telluride, Colo. This vein occurs in andesite and dips at 79 deg. Stopes were 225 to 250 ft. long, with 20-ft. pillars between. The Nevada-Massachusetts vein at Mill City, Nev., is 4.5 ft. wide. Harmony vein at Baker, Idaho, is 5 ft. wide. Cortez vein at Cortez, Nev., is 7 ft. wide. All these veins dip at angles steeper than 45 deg.

The Mount Hope vein, in New Jersey, is approximately 20 ft. wide, as is the Black Butte vein, in Lane County, Ore. Some shrinkage stoping has been done in the Michigan copper country on the Baltic lode, where the width of the orebody is approximately 20 ft. All of the instances mentioned are illustrations of shrinkage stopes whose long dimension is parallel to the vein.

In the wider orebodies, shrinkage stopes are laid out across the orebody, in which event pillars are left between them. At the Engels mine, in California, stopes are 70 to 100 ft. wide with pillars 20 ft. wide between. Stopes extend from footwall to hanging wall, about 100 ft. At the Homestake mine, in South Dakota, stopes and pillars are laid out at regular intervals from footwall to hanging wall. Stopes are 60 ft. wide and pillars 40 ft. wide. The maximum width of the orebody is 400 ft. Pillars are subsequently mined by square-setting. At Nacozari, Mexico, pillars 15 ft. wide are left between stopes. These pillars are later mined by undercutting the base and caving them. After all ore has been removed, the stope is filled with waste.

When considerable shrinkage stoping was being done at the Duluth mine, Cananea, Mexico, a rather unique and different pillar arrangement was used. Stopes were 60 ft. wide and 120 ft. across the strike of the orebody. Pillars between stopes were 40 ft. wide. These stopes were brought up from the 200 level to the 100 level. Above each of these stopes, on the 100 level, all of the irregular top of the orebody was first mined by square-sets, which were removed, leaving a large opening. The ore was firm and stood well except that occasionally a large slab or block was likely to come down. To prevent this, six or eight small circular pillars were left in the stope. These were 10 to 12 ft. in diameter and arranged in two rows parallel to the long dimension of the stope. They were effective in preventing large masses from breaking away, especially when the stope was breaking

[1]Information Circular 6293, U. S. Bureau of Mines. By Charles F. Jackson.

through into the open stope on the level above. After the broken ore had been drawn, the bases of these pillars were drilled and blasted. Their fall was effective in breaking them into small pieces and only a minor amount of blockholing was necessary at the chutes.

Sizes of pillars and stopes could be given for numerous other mines, but no best practice may be specifically noted as regard to their size. Local ground conditions must govern the final decision, as well as the plans by which the pillar will later be mined. In many mines the tendency has been to decrease the widths of pillars. This is especially true where shrinkage is used as an auxiliary method to caving, when the base of the pillar is undercut so that it can be simultaneously drawn with the broken ore.

Modifications of the Shrinkage Method.—Bureau of Mines Information Circular 6293[2] classifies the Alaska Juneau method of stoping as modified shrinkage. This is fitting, as the method is identical to large-scale shrinkage stoping except in the method of breaking ore. Instead of breaking with drill holes in the roof of the stope with a burden upon them of 4 or 5 ft. or more, coyote drifts are driven in the roof of the stope, and these drifts, having a burden of 35 ft., are blasted. Quarrying with coyote blasting is the cheapest surface method of breaking, and shrinkage with coyote blasting is the cheapest underground method of breaking ore.

The so-called Latouche system used at the Beatson mine, Alaska, is another modification of shrinkage. In this operation, drilling is done from chambers opened up in raises in the roof of the stope. From these chambers 20 to 40 holes are fanned out in all directions from the horizontal to a position pointing vertically downward. Before the final blasting, these holes are sprung. The raises are inclined and drilling chambers are opened at approximately 28-ft. vertical intervals. The block of ground broken by drilling in each chamber is 35x52 ft. and 28 ft. vertically. Stopes are 70 ft. wide, and pillars 30 ft. wide. Stopes are laid out transversely to the strike of the orebody, which has a maximum width of 340 ft. and is 800 ft. long. The pillars are mined by the same method after the stopes on either side have been completed.

At the Alaska Gastineau two methods of shrinkage are used, the typical method and a modified method. In the first, stopes vary in width from 40 to 120 ft. and are as much as 400 ft. long. The modified method is used in the softer ore. The bottom of the stope is undercut in the usual way and then a channel or cut 7 ft. high and 12 ft. wide is taken out along the footwall of the stope, after which similar cuts are taken across both ends of the stope. After one or more such cuts are taken out, the block within these cuts caves. Such a stope is 300 ft. long and from 90 to 100 ft. wide. The ore zone dips at an angle of 60 deg. Stopes are longitudinal to the strike of the ore zone and taken out for the entire width of the orebody. Pillars, 40 ft. wide, are left between stopes. From 70 to 85 per cent of the ore in such a stope caves. Vertical lifts are 200 ft.

In all of the modifications described, blockholing is one of the most important parts of the stoping operations.

Economics of Shrinkage Stoping.—A study of the economics of shrinkage stoping makes possible certain definite conclusions. The broadest and most general of such conclusions is that, other conditions being equal, the larger the orebody, the lower is the cost of stoping per ton and also the total cost of mining. However, there is not a uniform decrease in costs per ton as size is attained. Based on the nature and character of the development work, orebodies which are adaptable to shrinkage stoping can be divided into three main groups, which are (1) narrow veins, (2) wide veins, and (3) large orebodies. There is a definite relationship of group costs, the narrow veins having the highest cost per ton and the large orebodies the lowest. A more or less uniform relationship of cost to size obtains in each group, but when an orebody passes from Group 1 to Group 2, the cost may be slightly increased. This is explained by the development work necessary for extraction.

The veins considered in Group 1 are those which can be worked through a single longitudinal extraction drift which is usually driven in the vein. In this operation, that the cost per ton of ore of driving this drift, and the cost of the installation of chutes and of other development work, are inversely proportional to the width of the vein, is obvious. The stoping cost, however, is not proportional to the width of the vein. Take, for example, a 5-ft. and a 10-ft. vein. The latter will require more holes and more powder than the former, but never twice the amount, if ground conditions are equal. Similarly, other costs would have the same relationship. A curve of these costs starting with a 3-ft. vein would show a rapid drop of costs per ton per foot of increase in the width of the vein, but as width increases the curve would flatten out until it reached Group 2. In this group of wide veins are included veins which require crosscuts in addition to the longitudinal extraction drift. Additional raises are required, which add to the cost per ton, and no marked decrease in the cost per ton occurs until the width of the vein within this group increases in size, permitting still larger scale operations.

When the orebody increases to large dimensions, both horizontal and vertical, large-scale operations are possible. Examples of such shrinkage stoping are the Alaska Juneau, Alaska Gastineau, and the Beatson mines. Very low costs are obtained at all of these mines, but the total cost per ton at the Alaska Juneau is the lowest recorded underground mining cost in the world.

The relation of cost and size applies to no other stoping method as it does to shrinkage. Large horizontal and vertical dimensions are necessary for the application of caving methods, where low costs are expected, but with square-set stoping beyond certain limits larger size does not mean lower costs per ton. This is similarly true for the top-slice, open-stope, and the cut-and-fill methods. Extremely large size is advantageous for low costs only in caving and shrinkage.

General Considerations.—The breaking and drawing of ore in most methods of stoping usually complete the operation. With the cut-and-fill and square-set methods, filling is a necessary part of the operation and is carried on simultaneously with extraction. The top-slicing and caving methods are complete in themselves, but with shrinkage stoping, as well as with open stoping, cavities result, and these are a problem. If such cavities are large and numerous, they become liabilities. If the orebody which is mined by shrinkage is large and extends from level to level, vertical pillars are usually left between stopes and horizontal pillars under the level. Under these conditions, the adoption of the shrinkage method includes with it the decision as to the method of recovering the pillars and the disposition to be made respecting the cavities.

At the Homestake mine, the empty shrinkage stopes are filled and both the crown or level pillar and the stope pillars are mined by square-setting. The final result is identical to the results obtained by stoping by the cut-and-fill method. The same amount of fill is required and the same amount of square-setting is done. However, the shrinkage method has the advantage in that the intermediate steps are carried on more efficiently. The routine of the operations in the shrinkage stope are much simpler than in the cut-and-fill. In the former, they consist of drilling and blasting and drawing the overbreak; in the latter, one slice is removed, and the ore is shoveled into chutes and has to be carefully cleaned up. Chutes have to be carried up, waste must be delivered to the stope, this leveled off and flooring laid, after which drilling and blasting are repeated. Usually some time is lost in changing from one operation to another.

Summarizing the advantages of the shrinkage method over the cut-and-fill method, when used under the conditions described in the foregoing, we find that in shrinkage stopes:

1. Ore is broken more efficiently.
2. Shoveling is dispensed with.
3. Chutes and chute timbers are eliminated.

[2]By Charles F. Jackson.

4. Floors and the labor of placing them are unnecessary.

5. Because of (1), (2), (3), and (4) the cost per ton of stoping is considerably less.

6. No ore is lost in the fill.

The main advantage of the cut-and-fill method is that sorting is possible with a much mixed ore. In the use of shrinkage, the walls must stand sufficiently well to allow drawing the ore and subsequent filling without serious caving.

Comparison of the two methods from the viewpoint of safety is difficult. The cut-and-fill method allows the placing of cribs and stulls to hold up loose blocks and slabs. This is not possible usually in a shrinkage stope. On the other hand, such support is not often necessary in such a stope. The only men in a stope are the drillers. Mucking is seldom done, and chutes are not built up and waste does not have to be spread as in a cut-and-fill stope. The secret of keeping the roof in a safe condition in a shrinkage stope is close inspection, the barring down of all loose material, and rapid working. By working rapidly no time is permitted to elapse during which the roof might get bad, and in this way it is possible to stope ore by shrinkage which could not be mined by cut-and-fill. The removal of one slice across the roof of the stope is immediately followed by the removal of another. When this practice is followed, rather soft ores can be mined with comparatively little danger. The method is sometimes used in rather soft ground. This is often so where it is used as an auxiliary to the caving method.

In shrinkage stopes drillers are occasionally drawn into the ore and suffocated. Such accidents are not the fault of the method so much as of its improper application. Accidents of this kind can be avoided by a thorough understanding between the carman and the driller as to which part of the stope is to be drawn and which part to be drilled.

Filling the empty stope and mining the pillars by square-setting is the most common practice. In some operations, stopes are filled and the pillars are mined by top-slicing. This latter method has advantages and disadvantages. The latter are those which usually accompany top-slicing—namely, the possible destruction of valuable surface or the loss of possible ore and workings immediately over or adjacent to the slicing operations. Where such loss will not occur, top slicing has the advantage of being a cheaper method than square-setting. This advantage is greatest under conditions of soft ground, especially if pillars begin to take weight and crush.

At the Engels mine, pillars are recovered without filling stopes. This is done by drilling and blasting them by the ordinary methods and also at about half the cost per ton by using deep diamond-drill holes. In most mines ground conditions are not sufficiently good to permit of the removal of pillars without filling.

Where ground conditions are particularly good, stopes are filled and most of the pillar is mined by shrinkage. In this event, the so-called curtain pillar is left on each side of the stope. Under some conditions this pillar is cut down to a width of 5 ft. It keeps the waste from the adjacent stope from mixing with the ore while being mined and drawn. In no way is it a means of support for the roof of the stope. The ore in it is not recoverable.

Shrinkage as an Auxiliary Method.—Shrinkage stoping has two uses as an auxiliary to caving methods. The more important of these is as a means of delineating blocks to be caved, and when employed in this way the stope is known as a boundary shrinkage stope. Such stopes are often laid out to establish the boundary between waste and ore. At the Old Dominion, at Globe, Ariz., the entire orebody to be caved is freed from the surrounding waste by such stopes, and transverse stopes cut the orebody into different blocks of suitable size. At the Capote mine, at Cananea, parts of the orebody were thus freed from the waste. At Ray, Morenci, Inspiration, and Bisbee, use is made of shrinkage stopes in limiting the size of blocks and panels to be caved. Most of these stopes are little wider than a drift. In some instances it has been found unnecessary to carry the stope from the bottom of the orebody to the capping.

A different use of shrinkage as an auxiliary method is at Ray, where undercutting is accomplished by means of shrinkage stopes. The block to be caved is undercut by a series of parallel narrow stopes, which are widened at 35 ft. above the grizzly level so that they will connect and completely undercut the area.

Another somewhat different use of the method is to carry up stopes to the top of the orebody. Pillars between stopes are then cut off at the top and bottom, with the result that the pillar is drawn with the broken ore.

Conclusions.—Of the six principal methods of stoping, shrinkage stands third in point of production in the United States, being outranked only by caving and open-stope methods. Shrinkage costs vary so widely, because of local conditions, that to compare them is difficult. The modified shrinkage of Alaska Juneau produces the cheapest tonnage obtained by underground methods in the world. However, if an average cost could be established for the method, it would be higher than that of either caving or open stoping. Two reasons for the great variation in costs are, first, variation of the orebody in size and shape, and, second, whether or not filling is required.

The shrinkage method has many advantages over the higher-cost methods. It is my opinion that the most serious disadvantage is the fact that by its use large cavities are left. These sometimes become liabilities, with the result that they must be filled, which, of course, adds to the cost per ton of stoping and, therefore, partly reduces the advantage of using the shrinkage method. To leave substantial pillars and not fill stopes is possible, but valuable ore is lost or, at least, it is tied up and cannot be immediately extracted. Under certain conditions the latter method is the best. If several levels, one above the other, are worked, almost complete recovery of the pillars can be accomplished at the lowest cost. By filling the upper of these stopes, pillars can be recovered, and this waste then drawn from the upper stope into the next lower, these pillars recovered, and so on down to the lowest stope. Such procedure is based on the theory that the final caving of the upper areas will not be detrimental and on the expectation that gradual caving will result and thus prevent the possibility of a serious air blast. Such a method involves two important considerations: first, the original pillars must be made of sufficient size so as to prevent premature crushing, and, second, waste filling must be of a more or less selected character. If only hard blocky waste is used, it can be satisfactorily drawn for use below. Soft, talcy waste mixed with blocky material is likely to become consolidated to such an extent that it cannot be drawn from the upper stope and, therefore, cannot be used again.

On final analysis, methods of filling and pillar recovery are the most important part of the shrinkage method, inasmuch as they will determine the extent and degree of the advantage of the method over some other possible method.

Nickel in the Far North

Prairie Bay at low tide; the ore deposit is in the middle background near the tent at the extreme left

John Drybrough

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IN THE SUMMER of 1928 the Cyril Knight Prospecting Company, Ltd., of Toronto, dispatched a party of prospectors, Messrs. Rumsey, Rodstrom, Johnston, and Melvin, to explore the large area of favorable geology on the northwest shore of Hudson Bay. They made the hazardous 300-mile journey along the coast from Fort Churchill to Ranken Inlet in two 22-ft. canoes with outboard engines. Soon after their arrival, Johnston discovered a zone mineralized with pyrrhotite and chalcopyrite, and several claims were staked. The pyrrhotite proved to be nickeliferous, and, in the following season, the party dug several hundred feet of trenches and sampled the outcrop, which is about 200 ft. long and 75 ft. wide and lies on the high-tide mark.

The average of all the sampling was 0.56 per cent copper and 0.96 nickel, but along the edge of the outcrop next to the sea was a richer section averaging 1.25 per cent copper and 2.52 per cent nickel over a width of 9 ft., with the full width and length unknown. Of further interest was the fact that composite samples, containing 0.85 per cent copper and 1.75 nickel, assayed 0.08 oz. platinum per ton.

The function of the Cyril Knight Prospecting Company is, as its name indicates, prospecting, but the directors decided to carry out the next stage and contracted with the Smith & Travers Company, Ltd., Sudbury, Ont., for 2,000 ft. of diamond drilling. This was carried on in the summer of 1930, when I accompanied the party and spent nearly two months on the property.

Various difficulties, including caving and frozen ground, combined with the short season, prevented drilling more than 1,200 ft., but the results were satisfactory. Four holes cut commercial ore; one hole cut a narrow section of medium grade; and one was barren. A rough estimate of "possible ore" in the block from the drill holes to surface follows:

	Tons	Per Cent Cu	Per Cent Ni	Oz. Pt
Massive sulphide.....	30,000	1.28	9.25	0.20
Disseminated sulphide..	90,000	1.19	3.10	0.08
Totals.....	120,000	1.22	4.62	0.11

The grade of the massive sulphide is quite remarkable, so that, although the occurrence may not prove to be a very large one, it has commercial possibilities.

The property is situated at latitude 62 deg. 48 min. North, longitude 92 deg. 3 min., on a point on the north side of Ranken Inlet. This is about 250 miles south of the Arctic Circle, so that the deposit is one of the most northerly so far discovered. It is in the Canadian Pre-Cambrian Shield, with its characteristic topography, a glaciated peneplain of low relief, the maximum difference in elevation being not more than 150 ft. Ridges and valleys have a northwest-southeast trend, conforming to the general strike of the formations and shaped by the last glacial movement. Rock outcrops are numerous along the coast line, but inland they disappear beneath the drift.

Location of the deposit, at Ranken Inlet

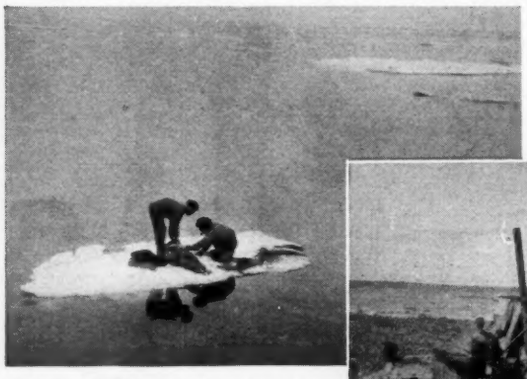
All the rocks probably belong to the Pre-Cambrian, and a tentative scale is given: *Intrusive:* lamprophyre and diabase dikes; granite; gabbro sills; and pyroxenite sill. *Extrusive:* upper volcanic, andesite lavas, and agglomerate. *Sedimentary:* tuff, conglomerate, and quartzite. *Extrusive:* lower volcanic, andesite, and basalt.

The volcanic and sedimentary members belong to the Keewatin or a contemporaneous period. The interrelation of the granite, gabbro, and pyroxenite is not known. They are older than the lamprophyre and younger than the upper volcanic.

Geological investigations disclosed the fact that the volcanic and sedimentary rocks have been folded into a broad syncline, its axis pitching flatly to the southeast. The lower volcanic-sedimentary contact runs up Melvin Bay, southwest of the property, circles to the north and northeast, and down the far side of Prairie Bay to the northeast. The contact of the sedimentary and upper volcanic curves across the end of the point; intrusive into the north side of this contact is a pyroxenite sill, traceable for 4,000 ft. west from Prairie Bay. Whether it follows the contact around to the south is not known. It does not outcrop on any of the islands to the east. The thickness is 200 to 300 ft. and it dips, according to drill-hole intersections, about 50 deg. south, but it is thought to steepen toward the west and flatten toward the east. It is

overlain by volcanic diorite and underlain by banded tuff which gives way to thin-bedded quartzite at shallow depth. The orebody is a sulphide replacement in a flank of a "bulge" or "roll" of the footwall of the sill.

Originally composed of pyroxene and plagioclase with accessory magnetite and pyrite, the pyroxenite is now a serpentine rock with carbonate, magnetite, and pyrite. Along shearing and jointing, further alteration to soapstone and talc schist has occurred. The ore consists of massive and disseminated pyrrhotite with minor amounts of chalcopyrite and pyrite. No nickel mineral has been recognized, but it probably exists as pentlandite (Fe, Ni)S, very finely divided or in solid solution in the pyrrhotite. Available data are insufficient to suggest a relationship for the platinum with either copper or nickel, and the same holds for palladium, of which a small amount is known to be present.



The commissary department at work: Eskimo guides cutting up a seal

Diamond-drill set-up, showing improvised water heater

The sequence of deposition was pyrite first, later replaced by pyrrhotite accompanied by the nickel mineral and part of the chalcopyrite, the remainder being deposited later. The sulphides are inclosed in soapstone, and considerable actinolite is present in the massive sulphide zone.

A lens of fairly clean massive sulphide occurs in the sill at or close to the footwall. Its surface projection lies beneath the high-tide line under the shore drift and does not outcrop. On the hanging-wall side of this lens is a zone of disseminated ore, part of which shows on the edge of the mineralized outcrop. Its length is about 250 ft. and the average widths, according to the drilling, are 8.0 ft. massive, 18.3 ft. disseminated, and 24 ft. combined. The body rakes flatly to the west, with that end of it not delimited. The dip is 50 deg. south.

A second disseminated sulphide zone occurs about 40 ft. south, but it is below commercial grade—0.75 per cent copper and 0.95 per cent nickel. The footwall of the sill is heavily drift-covered, but the dip needle indicates one or two places which deserve drilling.

An extraordinary feature of the drilling was the frozen condition of the ground, which persisted to the limit of the drilling, 220 ft. vertical. Strong brine solutions froze readily in the holes. Circulating drill water was drawn from the sea, and a depth of 50 ft. could be attained without trouble, but, below that, the water would be chilled by overlong contact with the frozen rock, and it would freeze, seizing bit and rods as firmly as if cemented. Twisting and sledging were of no avail and generally resulted only in broken rod connections. Resources were limited, but the foreman devised a reaming bit from a length of 1¼-in. pipe and some chip diamonds and enlarged the hole around the seized string, which was then released and disconnected joint by joint.

Without heating the circulating water no hole could have been completed. The drill was gasoline-driven, so a heater was improvised from a gasoline drum, with pieces of drill rod for grate and 1-in. pipe for a water coil, and a supply of coal was brought from a cache, 60 miles away. It had to be conserved most carefully, first because there was only 2,500 lb. available, and second, because it cost $106 per ton at the cache. This worked satisfactorily, but as fall approached and the sea temperature fell, the heater was overtaxed and the last hole was completed only with great difficulty. Pump trouble or any other stoppage of the circulating water had to be righted immediately or removal of the bit from the hole would have been imperative. It could not be left for more than twenty minutes or it would be frozen in, however warm the water had been before the stoppages. Another bad feature was the frost-heaved and fractured condition of the surface zone, about 25 ft. deep. The supply of cement was limited, casing was expensive, and its installation slow.

Sludge samples were not reliable, for much material was lost and frozen in fissures in the rock. Fortunately, core recovery was fairly good.

The experience will be most valuable. With ample provision for heating the circulating water and of cement for the fractured surface zone, drilling should be carried on as easily as in more temperate climates. Much credit is due to the foreman and setter, J. Cyr, by whose ingenuity and resource the work was accomplished.

The property seems remote, but it is almost as close to Liverpool, England, as is Montreal, 2,850 miles compared with 2,750 miles, and it is 3,100 miles from New York. In this region the season of navigation lasts about three months from the middle of July. The north shores of Ranken Inlet are studded with reefs and shoals, but good channels and anchorages exist. Deep water is less than a half mile from the deposit, and a small sheltered bay 2,000 ft. south has possibilities as a harbor for vessels of moderate tonnage.

At present, Fort Churchill, the terminal of the Hudson Bay Railway, is the handiest point of access to the area, and the drilling crew were transported from there in a 45-ft. steel launch. However, the west coast of Hudson Bay is not safe for small boats. For miles out, the water is shallow, reefs are plentiful, no safe anchorages are to be found for 200 miles north of Fort Churchill, and bad storms are common. The outward journey took five days, but the return took twenty-three, owing to three prolonged storms. The last blew the launch ashore from its anchorage and so damaged it that the party had to return by a Hudson's Bay Company boat. A larger vessel could have ridden out the storm at sea without danger.

Operation of mines in the North under sub-Arctic climatic conditions is bound to involve problems not encountered elsewhere. The frozen rock will affect underground air and water transmission and drilling, but there should be no pumping problem in spite of the proximity to the sea. The region is treeless, and all timber, as well as fuel and provisions, will have to be brought in by boat during the short season of navigation. Though the deposit is not large the grade is good, and profitable operation is indicated.

Life for the operating force will be quiet and visitors infrequent, but the radio has transformed conditions in the North, and the job, with its problems, will be interesting. Mines have been worked farther North—the Greenland cryolite deposit and the Spitzbergen coal mines, for example—so it is not unique. For the hunter, the trapper, and the prospecter this virgin country has much attraction; I have never spent a more interesting or enjoyable season in the field.

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Standardized Sieving Methods

Lack of uniformity in sieving data has always handicapped milling operations. The method here presented promises consistent, reliable results and easy and rapid manipulation.

Alan Probert

SIEVE ANALYSES, when properly performed, add valuable information to the store of knowledge accumulated by the metallurgical engineer. Most of the generally accepted methods of test sieving unfortunately exhibit a tendency toward cumulative errors. This is readily understood in view of the fact that undersize from any one sieve makes up the entire feed for the next size. Thus an error in percentage of undersize at any point in the series is transmitted to subsequent fractions. Particles larger than the apertures in the wire cloth cannot pass into the undersize unless forcibly impelled—as for instance by the use of washers on the sieve during the sieving procedure. The conclusion cannot be drawn, however, that all fragments inferior in size will pass through the openings. Retention of material fine enough properly to be classed as undersize necessarily increases the apparent percentage of oversize and reduces the proportion of undersize. Each recurrence of this error reduces the accuracy of the determination, making a serious cumulative effect when the range of sizes is extensive. In consideration of the widespread use of the results of sieve analyses, greater emphasis on the means of securing the data appears advisable in order that increased reliability may be obtained.

The U. S. Bureau of Mines, the Massachusetts Institute of Technology, the Milling Committee of the A.I.M.E., and the University of California, in the years 1922-24 co-operated in research to develop a standard method of sieving for the mining industry. Notable among the excellent proposals resulting from this investigation was the procedure advocated by John Gross, of the U. S. Bureau of Mines. A unique feature of the suggested procedure, in its application to the metallurgical field, was its dependence on a time-percentage weight end point similar to that established by the U. S. Bureau of Standards for use in making size specifications for cement and pulverized coal. It also included the reliable wet-sieving operation for desliming the sample prior to the dry sieving of the coarser portion. Losses were made self-evident and were easily allocated by the provision for weighting oversize as well as undersize after separate manipulation of the various sieves. The accuracy of the results can be judged by comparing duplicate fractionations of the same sample. Duplicate sieve analyses by the Gross method check to 0.1 per cent ($\pm$ 0.05 per cent). Because the fundamental nature of sieving precludes the establishing of an absolute standard in the same manner as in chemical analysis, duplicate sieving of the same sample must suffice as the only indication of precision. The high degree of accuracy inherent in the Gross standard hand-sieving procedure is rarely a justification for the extremely tedious work required by the separate manipulation of the sieves.

At most ore-dressing plants one of two generally accepted procedures is followed in test sieving, irrespective of whether the vibratory motion is imparted to the sieves mechanically or by hand. The first is strictly dry sieving with no attempt to remove the extremely fine particles adhering to, and cemented upon, the surfaces of the larger fragments. In the other common method the pulp is first washed on the finest sieve—generally 200 mesh—whereby the almost infinitely fine material passes through with the water as a preliminary undersize. The dried oversize is then sieved separately. A much cleaner separation is made possible by the second method, and the additional time consumed is more than justified by the increased accuracy. Whether the entire sample is sifted dry or a preliminary wet-splitting on 200 mesh is used, a definite period of shaking the sieves is generally maintained. The results may in some degree be considered comparable, when similar ores with approximately the same degree of comminution are fractionated. Unless the sieving is actually carried to some definite end point, however, the results will have no absolute value.

The purpose of this article is to propose for consideration a method that has been found reliable and accurate, and which consumes a reasonable amount of time. This procedure is not claimed to be a substitute for the method just described as applicable when the utmost precision is essential, but it has a wide application in the metallurgical industry. Its limit of error is of the same magnitude as the limits common in assaying and sampling. The results check closely with those obtained by the accurate method of hand sieving, as will be demonstrated, and the data can be obtained in time to be used for controlling the plant, instead of being of historical interest only. The method embodies the time-percentage weight end point for dry sieving, which is performed in standard manner on the Tyler Rotap, and the wet split on 200 mesh is incorporated to add dependability. After thorough trial over a period of three years it is set forth as possessing advantages that cannot be claimed for other methods.

Data presented in Table I demonstrate the close agreement between results obtained by hand sieving and the method advocated, which for lack of other designation will be referred to as the standard procedure.

The first step, preparation of the sample, requires the same diligent care used in preparing samples for assaying. A convenient size is a sample of one kilogram; the fractions resulting from such a sample are sufficient for subsequent work, and the end-point weight, 0.1 per cent of the original sample, is one gram. The sample is deslimed on the sieve of smallest aperture, ordinarily 200 mesh (0.074 mm.). Careful manipulation in wet sieving minimizes losses. After wetting down the pulp in a pan, the suspension of fine particles is decanted through the sieve into a suitable container. This procedure is repeated two or three times until enough slime is eliminated to permit washing part of the coarser aggregate onto the sieve—sufficient to spread a thin layer over the entire area. Water is poured in, and the frame tapped repeatedly with the hand to cause a slight movement of the particles. The water passing through the sieve will soon appear relatively clear. Oversize is washed into a pan for drying and the procedure repeated until the entire sample has been treated. Brushing, rubbing with the fingers, or washing with a high-pressure jet contribute to inaccuracy and should be assiduously avoided. Wet split on 200 mesh is to be viewed as an aid to, and not a substitute for, the 200-mesh separation. Thorough performance of the wet sieving has a marked effect on the time required for finishing the dry sieving. Effort expended in making a fairly clean wet split is well rewarded in time saved later.

The wet fractions are dried and weighed separately; the sum of the two weights may be taken as the original

weight for subsequent calculations, thus eliminating small errors such as the discrepancy caused by small amounts of soluble constituents. After drying and weighing, the wet oversize is ready for the preliminary sieving, the purpose of which is to save time by preparing the material for the actual end-point sieving. Before entering into a discussion of the preliminary sieving procedure, however, the end point must be defined.

In this procedure the basic premise is that under well-defined sieving conditions only a certain designated amount of undersize must be produced by any sieve in a specified time. Use of the Rotap standardizes the motion of the sieve and the conditions of shaking. Amount of undersize and the duration have been arbitrarily chosen as 0.1 per cent in one minute. The limit of accuracy is, therefore, ± 0.1 per cent, with a maximum error of 0.2 per cent.

After the preliminary sieving of the wet split oversize, all sieves are thoroughly cleaned by brushing both sides to remove dust and particles held in the meshes. From this point the sieves are treated individually, the oversize being returned to the cleaned sieve and shaken for one minute at a time. The end point is first determined on the coarsest sieve, so that the undersize produced may be added to the oversize of the next smaller sieve in order. When the undersize produced in one minute of sieving is less than 0.1 per cent (one gram for an original sample of one kilogram) the end point has been reached. If a large number of apertures in the wire cloth contain fragments of ore, the sieve is said to be blinded. Should the sieve show evidence of such blinding after the end point has been reached, the result must be checked by another one-minute test, as a blind sieve frequently gives a false end point.

The preliminary sieving schedule should be such that the end point is reached in two or three one-minute periods. Fifteen-minute periods in the Rotap, followed by thorough brushing, give satisfactory results with many ores.

The following schedule will be found useful:

3 to 28 mesh, inclusive: one 15-minute period

35 to 48 mesh, inclusive: two 15-minute periods

65 to 200 mesh, inclusive: three 15-minute periods

Table I—Results of Hand Sieving and Standard Mechanical Method

Mesh	Hand Sieving, Gross Method Per Cent Weight	Mechanical Sieving, Standard Method Per Cent Weight
Minus 200	55.4	55.2
Minus 150, plus 200	10.9	11.3
Minus 100, plus 150	9.7	9.8
Minus 65, plus 100	12.0	12.3
Minus 48, plus 65	8.3	7.7
Plus 48	3.7	3.7

When this method is applied to an ore for the first time, a record should be made of the number of one-minute periods required by each sieve for finishing by end points, after a preliminary sieving according to such a time schedule. The duration may then be modified to obtain the end point in satisfactory time. The application of the end point establishes the finality of the work regardless of whether or not the preliminary shaking has been designed for greatest economy in time.

To recapitulate, the procedure described is as follows: (1) One kilogram is weighed out and pulped with water. (2) The sample is divided into a preliminary undersize and an oversize for dry sieving by wet split on 200-mesh sieve. Combined weight of these two products is used as basis for the calculated percentages of sizes. (3) A nest of sieves is made up including the sieve used in making the wet split. The dried oversize is given a preliminary sieving in the Rotap. More preliminary sieving is required for the finer sieves than for the coarser ones. A tentative schedule for this work has been presented. (4) The end point determined for each sieve by one-minute periods in the Rotap is represented by 0.1 per cent of undersize produced in one minute of Rotap sieving. When the end point is reached the sieve is examined. If blinded, it is cleaned and the one-minute end point repeated. A blind sieve has frequently been found to give a false end point. (5) The minus-200 mesh material from the dry sieving is added to that obtained in the wet split for weighing.

Adoption of a well-defined standard method of sieving would greatly increase the value of published data and permit comparisons between mills treating—in various ways—ores of entirely different characteristics. At present such comparisons are practically without meaning. In plants where some particular method has been adopted as standard for certain operations, relative results are obtained. By comparing the results with those obtained by a method based on end points, a correction curve may be plotted to convert the data to a comparative basis. Sinter and other friable materials cannot be sized any more accurately by this method than by others. The end point for readily abraded products is obviously of no value. Wet sieving practice is unsuited for chemicals, cement, and similar materials. Ore-dressing plants offer such a broad field that a method adapted particularly to concenration problems is justified.

Table II—Results of Sieve Analysis, by Various Methods, on the Same Sample of Ore

Mesh Size	A	B	C	D	E	F	G	H
Minus 200	44.1	47.3	48.1	64.7	65.5	66.2	67.6	67.6
Minus 150, plus 200	23.9	24.4	27.0	14.6	15.0	15.2	14.5	14.4
Minus 100, plus 150	8.5	8.6	8.1	6.6	5.8	6.2	5.8	5.9
Minus 65, plus 100	18.6	15.7	13.4	10.2	10.2	9.6	9.3	9.3
Minus 48, plus 65	3.9	3.0	2.8	3.1	2.7	2.5	2.5	2.5
Plus 48	1.0	1.0	0.6	0.8	0.8	0.3	0.3	0.3

A: 15 minutes dry sieving—no wet split.
B: 30 minutes dry sieving—no wet split.
C: 60 minutes dry sieving—no wet split.
D: Deslimed wet, oversize dry sieved for 15 minutes.
E: Deslimed wet, oversize dry sieved for 30 minutes.
F: Deslimed wet, oversize dry sieved for 45 minutes.
G: Deslimed wet, preliminary sieving according to schedule, all end pointed.
H: Deslimed wet, preliminary sieving according to schedule, all end pointed.

Conclusions and Summary

1. A rapid method of sieving that will reproduce results within definite limits is a much felt need in the mineral industry.

2. In most sieving procedures a tendency toward cumulative errors exists. A 100 per cent separation at any sieve size is seldom obtained. The use of washers on the sieve prevents blinding of the apertures, but such a procedure forces oversize particles through the openings. When sieving is followed by microscopic examination the washers have a serious effect.

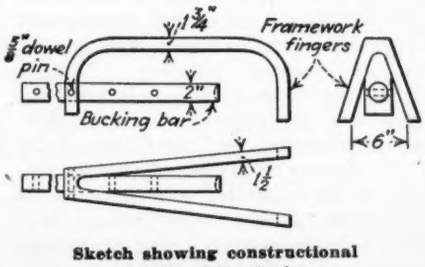
3. The industry would be benefited by the publication of results substantiated by a standard method rather than those based on random procedures of dubious accuracy.

4. A method has been described that meets the specifications; and adequate proof of the reliability of this method has been presented.

■

Adjustable Bucking Bar

IN CONSTRUCTION work, and where a large amount of riveting of the same kind is to be done, an adjustable bucking bar may be used to great advantage, writes R. M. Thomas, of Denver, Colo. A bar is made of 2-in. round steel of a length depending upon the work at hand. Framework fingers, 1½x1¾ in., with ends bent to fit over the frame or rivet end, hold the bar rigid. The bucking bar slides through a hole in the frame and is held in place by a ⅝-in. dowel pin. The distance between the fingers at the rivet end should be 6 in., in order to permit proper handling of rivet and air hammer.



Sketch showing constructional details of the device

Selective Lead-Zinc Flotation at Kimberley, B. C.

Second of a series of articles on the mining, milling and metallurgical operations of the Consolidated Mining & Smelting Company of Canada, Ltd.

George J. Young

Associate Editor, San Francisco

SULLIVAN ore is crushed near the portal of the mine adit before transporting it to the concentrator. The first unit of the coarse-crushing plant is a receiving bin, with a capacity of 1,200 "live" tons. Mine ore is reduced by two jaw crushers, a 42x48-in. Allis-Chalmers and a 36x42-in. Buchanan, the discharge passing over bar grizzlies spaced at 1¾ in. Oversize passes over picking belts from which rich lead ore and waste rock are removed. These belts discharge into three Traylor No. 10 gyratories, the grizzly undersize and the gyratory product being delivered to a shuttle conveyor on top of a 2,300 live-ton storage bin. From this, a train of air-dump cars transports the ore 2½ miles to the fine-crushing plant at the concentrator. Additional sorting-belt capacity is being provided at the coarse-crushing plant.

The fine-crushing unit and flotation concentrator are planned to handle 6,000 tons a day. The great density of the ore must be kept in mind when considering the number of units required in fine crushing and in fine grinding. Although the ore is a massive sulphide, it breaks down readily. Present fine-crushing from a maximum size coarse-crusher product of 3 in. yields material 2 per cent of which is retained on a ½-in. screen.

The receiving bin, of reinforced concrete, 1,800 tons' capacity, delivers to two conveyors, which discharge on to a cross conveyor delivering to a surge bin and a 4x11-ft. Hum-mer screen, which bypasses fines. Oversize is delivered to a 7-ft. Symons cone crusher set to ⅝ in., driven through a Texrope by a 200-hp. motor. Screen undersize and cone-crusher product are conveyed to a surge bin, from which roll feeders deliver to four 4x5-ft. Hum-mer screens, undersize being conveyed to the mill bins, and oversize to two sets of Allis-Chalmers 74x20-in. Garfield-type rolls. The roll product is delivered directly to the mill-bin conveyors. Arrangements are completed for operating the rolls in closed circuit with the screens, so that a minus-⅜-in. screen product can be obtained.

At the time I visited the plant a dust retrieving system, designed by George E. Lynch, was being installed. This provides an exhaust fan that places a cylindrical dust-catching chamber under vacuum, this connecting with a flue system providing dust take-offs at Symons crushers, surge bin, screens, and rolls. A cyclone removes coarse dust before the dust stream enters the spray tank. The system is the development of installations made by Mr. Lynch in Arizona and elsewhere.

Uniformity in flotation mill feed is obtained by the mine requirement of uniform grade, by bedding in the coarse-crusher shipping bin, by bedding in the receiving bin, and by final bedding of the fine ore in the mill bins, of which there are two—a 5,000-ton catenary steel structure and a 4,800-ton reinforced concrete bin, both served by Link-Belt bedding machines.

Fine grinding is accomplished in three stages. Weightometers control the initial feed. The first stage consists of six 10-ft. by 48-in. Hardinge ball mills, which deliver to three Dorr quadruplex classifiers (16x20 ft.). Each primary mill is provided with a trommel screen with inner spiral discharges to remove chips. Primary classifiers remove a finished product that is delivered to the flotation surge tank. The sand discharge is delivered to the second-stage grinders—four 8-ft. by 48-in. and four 10-ft. by 48-in. Hardinge mills (trommel screen discharges)—which are operated in closed circuit with the secondary classifiers. Ten Wilfley sand pumps (four 8 in. and six 6 in.) return the ball-mill discharge to two distributors, which divide the product among 21 secondary classifiers, ten being standard Dorrs (6x25 ft.), and eleven Akins classifiers (72-in. spirals). Sand product is returned directly to the secondary grinders. Classifier overflows are pumped by eight Wilfleys to distributors feeding the tertiary classifiers. Classifier overflows go directly to the flotation surge tanks joining the primary classifier overflow, which is pumped by two Wilfleys into the surge tanks. Sand discharge from tertiary classifiers, ground by two 8-ft. by 48-in. Hardinge mills, is returned by pumps to the tertiary classifiers. To control pulp dilution, part of the tertiary classifier overflow is deflected to two 18-ft. Genter thickeners, which remove part of the water and provide a thickened intake for a battery of three pumps delivering to the flotation surge tanks.

Screen analyses of the successive stages in the grinding circuit are given in Table I. A feature of the fine grinding is the maintenance of a large circulating load in the secondary and tertiary grinds. Three-inch cast-iron balls are used on the primary mills, and 2½-in. balls on the secondary and tertiary grinders. Rubber pipe bends are used for handling coarse grind, and launders are rubber lined, with soft rubber on a single thickness of canvas to hold the nails, the lining being ⅜ in. thick. Pipe bends have lasted 1½ years, and launders are in service after five or six years. Kimberley pioneered in the use of rubber linings, and they have proved to be most efficient. Pumps, of white cast iron, are manufactured at Trail.

Ball mills are lined with manganese steel, a material more and more popular to resist abrasive wear. Manganese-steel cams and rollers are used on the

Table I—Screen Analyses, Grinding Circuit

Grinding Stages	Screen Size, Cumulative Percentage			
	Plus 75	Plus 150	Plus 200	Minus 200
Primary mill discharge	4.7	43.9	54.9	45.1
Classifier sand	7.3	61.4	72.8	27.2
Classifier overflow	...	5.2	11.4	88.6
Secondary 8-ft. mill discharge	0.7	49.1	66.5	33.5
Classifier sand	0.8	52.5	70.7	29.3
Dorr classifier overflow	...	9.7	18.8	81.2
10-ft. mills discharge	0.5	38.8	56.8	43.2
Akins sand	0.8	42.1	61.6	38.4
Akins overflow	...	6.5	13.4	86.6
Tertiary 8-ft. mills	...	23.2	42.3	57.7
Dorr sand	...	30.9	53.5	46.5
Dorr overflow	...	3.7	9.5	90.5
Final tailing	...	8.7	19.4	80.6

Primary mills: Pulp contains 69 per cent solids.
Classifier overflows: Pulp contains 39 per cent solids.

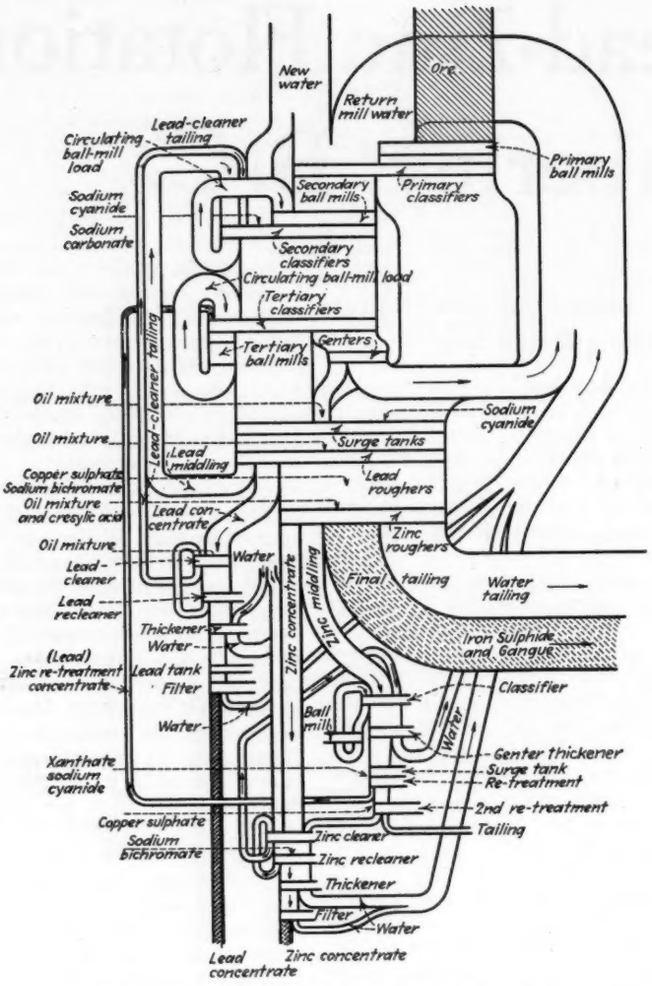


Fig. 1—Diagrammatic flow sheet of concentrator

Dorr classifiers. Ball-mill liner wear is 0.1 lb. per ton (two years' life); ball consumption, 1.3 lb.; fine-crushing rolls, 0.046 lb. per ton; Symons cone, 0.02 lb. per ton, and Tyler ton-cap screens (special alloy) last a month. Bolts for retaining ball-mill liners are placed in the "lifter bars" instead of between the bars. Power consumption on the Symons crusher is 0.48 kw.-hr. per ton; on the rolls 0.6 kw.-hr. per ton, and on the fine-grinding circuit 20 to 21 kw.-hr. per ton. Primary ball mills average 230 hp.; secondaries, 140 hp.; tertiaries, 130 hp. A bonus, based on the fineness of the final tailing with adjustments made necessary by "insoluble" content of ore, is given to the men in the grinding section.

Flotation Practice

Sullivan ore is an intimate mixture of sulphides associated with minor amounts of gangue minerals. Fine grinding is an essential preliminary to the liberation of the individual mineral particles. Grinding practice already outlined represents progression in the direction of finer and finer grinding, and it is not unlikely that this feature of the Sullivan concentrator will be extended further. At present the ore reaches the surge tanks feeding the flotation division with the approximate screen analysis of 4 per cent on 150 mesh; 6 per cent on 200 mesh; and 90 per cent through 200 mesh. The oil mixture, sodium carbonate (no lime is used at all), and sodium cyanide have been added in amounts approximating 0.3 lb., 2.75 lb., and 0.09 lb. per ton, respectively. These have been mixed with the ore pulp, and receive a final mixing in the surge tanks, of which there are two, one 30 ft. and the other 20 ft. in diameter, both equipped with stirrers. Both perform the functions of conditioners and distributors.

Fig. 1 indicates the flotation treatment and the places where reagents are added. The assay of the present feed, in percentages, is lead, 9; zinc, 7½; iron, 36; "insoluble," 14. Expressed quantitatively, the flotation requirement is the separation of 700 tons of lead concentrate; 825 tons of zinc concentrate, and the elimination to tailing ponds of about 3,975 tons of iron sulphide and gangue, based upon the 5,500 tons of ore treated.

The Consolidated company pioneered in selective flotation on a large scale, and its Sullivan concentrator has served as a standard of what can be accomplished with a difficult ore. Its flotation practice has been improved almost continuously, and the present status is of considerable interest. The broad features of the separation are as follows: a separation into three products by the "lead rougher cells"—(1) lead middling, returned to the secondary classifiers; (2) lead concentrate, which passes into the lead cleaning circuit; and (3) a "tailing" product, which is activated with copper sulphate and sodium bichromate and sent to the zinc rougher cells. These latter also produce three products—(1) zinc middling, which goes to a separate regrinding and cleaning circuit; (2) zinc concentrate, which goes to a cleaning circuit; and (3) a tailing that goes to waste. Alkalinity of the circuits is maintained with sodium carbonate, and sodium cyanide is used as a zinc and iron depressant.

The lead roughers consist of eight sub-A and two standard M.S. machines, all equipped with 24-in. impellers. Twelve cells of the sub-A machines are used for the separation of the lead concentrate (30 to 32 per cent lead), and four cells for separating the middling product (12 per cent lead). On the standard machines, fourteen cells are used for rougher concentrate and four for middling. Most of the cells are equipped with motor-Texrope drives, a single motor for each pair of cells. Additional oil mixture is added to the sixth and fifteenth cells of the lead roughers, making the total 0.41 lb. per ton for the lead-rougher circuit. Middling is pumped back to the distributors feeding the secondary classifiers. Rougher concentrate is sent to four eight-cell, spitz-type M.S. machines, which separate a 53 per cent lead concentrate for the lead recleaner cells, and a tailing (18 per cent lead) that is returned to the secondary classifiers. The lead concentrate is recleaned in two eight-cell machines of the same type, producing a concentrate carrying 70 per cent lead (4½ per cent zinc, 5½ per cent iron, 22 to 24 oz. silver, and less than 1 per cent "insoluble") and a tailing (45 per cent lead) which is pumped back to the lead cleaner cell. Final lead concentrate is sent to a 50-ft. Dorr thickener, the thickened product going to a 20-ft. lead-stock tank supplying three American (6-ft. 6-leaf) filters and one 8x12-ft. Oliver filter. The concentrate (9½ per cent moisture) is delivered to railroad cars.

Lead-rougher tailing approximates 1.5 per cent lead and 8 per cent zinc. To this is added 0.18 lb. sodium bichromate and 0.65 lb. copper sulphate per ton of original feed. In the zinc rougher machines, at the seventh cell, an additional 0.05 lb. per ton of oil mixture, plus 0.01 lb. of cresylic acid, is added. Zinc roughers consist of three eighteen-cell standard M.S. machines and six sixteen-cell sub-aërated M.S. machines. Rougher tailing, containing 0.65 per cent lead, 0.8 per cent zinc, and

43.8 per cent iron, is sent to waste. Zinc rougher concentrate contains 36 per cent zinc.

On the first zinc cleaner, an eighteen-cell standard M.S. machine, the grade is raised to 42 per cent zinc, the tailing being sent to the zinc-middling circuit. Zinc concentrate is sent to the recleaner, to which is also added 0.02 lb. per ton sodium bichromate and diluting water. Recleaner products comprise a tailing, returned to the cleaner machine, and a 50 per cent zinc concentrate, which is sent to a 50-ft. Dorr thickener and then to a 20-ft. stock tank that supplies three six-leaf American filters. Concentrate is dewatered to 10 per cent moisture. The final product contains 50 per cent zinc, 11 per cent iron, and 1 per cent "insoluble."

Zinc middling from the zinc roughers, plus the tailing from the zinc cleaner, are sent to an Akins classifier (72-in. spiral), in closed circuit with an 8-ft. by 48-in. Hardinge ball mill. Overflow is dewatered by three 15-ft. Genter thickeners, the thickened sludge being reamed in a 20-ft. surge tank that supplies two Wilfley pumps delivering to an eighteen-cell standard M.S. cell. Reagents, consisting of 0.05 lb. sodium cyanide per ton of original feed ($\frac{1}{4}$ lb. per ton of re-treatment feed) and 0.03 lb. xanthate (0.15 lb. per ton re-treatment feed), are added to the feed of the re-treatment cell. The concentrate, containing 8 per cent lead, is returned to the secondary classifiers of the secondary grind, and the tailing to the second re-treatment machine (eighteen-cell, standard M.S.), receiving 0.10 lb. copper sulphate per ton of original feed (0.5 lb. per ton of re-treatment feed). Zinc concentrate from the second re-treatment machine, containing 2.6 per cent lead and 28 per cent zinc, is sent to the zinc cleaner. The tailing, 1.1 per cent lead and 1.02 per cent zinc, is sent to waste.

Concentration ratio is 4.38 tons of feed to one ton of concentrate. Pulp density of rougher flotation circuits is 36 to 37 per cent solids; of tailing, 26 to 27 per cent solids. Temperature of lead rougher pulp is 28 deg. C.; of zinc rougher pulp, 30 deg.; of zinc and lead cleaner and lead recleaner circuits, 30 deg.; of zinc recleaner circuit, 35 deg.

For pulp-heating requirements, 12,000 lb. of live steam per hour is required. Total power ranges from 22 to 24 kw.-hr. per ton. Excluding repair men, the number of men per shift in the fine-grinding and flotation sections is 24. Total reagents per ton of ore, in pounds, are: Sodium carbonate, 2.75; sodium cyanide, 0.14; oil mixture, 0.48; cresylic acid, 0.1; sodium bichromate, 0.17; copper sulphate, 0.66; and xanthate, 0.03. These amounts vary with the metal content and feed conditions. Oil mixture consists of 60 per cent water-gas tar, 20 per cent A.T.P. No. 7 creosote, and 20 per cent cresylic acid.

Ore feed is controlled by weightometers. From the fine-ore conveyor an automatic bucket sample, equaling one ten-thousandth, or 1,200 lb., is taken each 24 hours. Tailing samples are taken each shift, and other samples wherever necessary. Examination of various mill products by the microscope is made at regular intervals. For this purpose microscopes are placed at tables and pulps are examined on glass slides. Results of rapid methods of analysis from a near-by chemical laboratory enable quick comparisons to be made between the indications of the microscope and the laboratory percentages. The microscope thus gives an accurate and quick check on pulps.

As a further aid to quick adjustment of flotation reagents, a pilot Wifley table takes the tailing from the lead recleaner cell, and from the width of the galena band the amount of oil mixture is determined. If galena is more prevalent than usual, more oil is added, and vice versa. Subaëration machines require less oil reagent than the standard M.S. machines. Another pilot Wifley table is also available for testing the zinc-rougher feed. Microscopic examination of the final zinc froth gives accurate indication of grade.

Concentration ratios have varied. At the beginning of milling operations at Kimberley, late in 1923, the ratio was 2.9, indicating a rich ore. In the first half of 1930, the ratio was 4.38. Fig. 2 shows the annual ratios. The recent ratio indicates that a much lower grade of ore is being treated than in 1923 or 1926. Ratio of lead concentrate to zinc concentrate tends to approach 1, the lowest ratio, 0.62, being in 1923, and the highest, 1.13, in 1927. This figure depends principally upon the relative amounts of lead and zinc sulphides in the ore.

Probably the best indication of the increase of flotation efficiency is afforded by the grade of the various concentrates and the percentages of the elimination of iron in the sulphide tailing. Grade of lead concentrate increased from 63.5 per cent lead in 1923 to 69.5 per cent lead in 1930; grade of zinc concentrate, from 38.4 per cent zinc in 1923 to 49.9 per cent in 1930; elimination of iron was 85 per cent in 1923 and 94.4 per cent in 1930. In 1923 a unit weight of metal (lead and zinc) carried with it 0.506 unit of sulphur; in 1928, 0.397 unit of sulphur. Not only was the metallurgical practice directed to the production of cleaner concentrates, but it also had an important objective in eliminating the maximum amount of sulphur in the tailing. Recovery of lead in the lead concentrate is 90 per cent, with an additional 4.7 in the zinc concentrate, or a total of 94.7 per cent. Recovery of zinc in the zinc concentrate is 83.5 per cent, with an additional 7.5 per cent in the lead concentrate, making a total of 91 per cent. Considering the nature of the ore, these results reflect great credit on the technical staff.

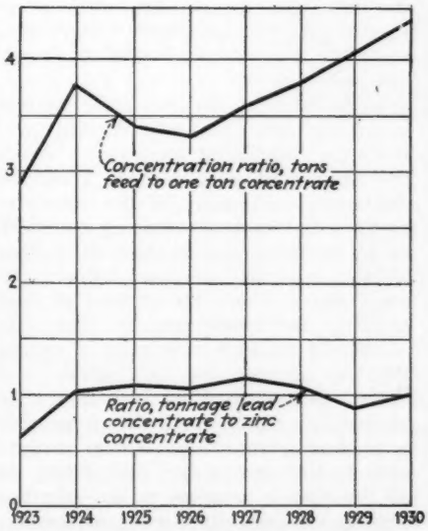


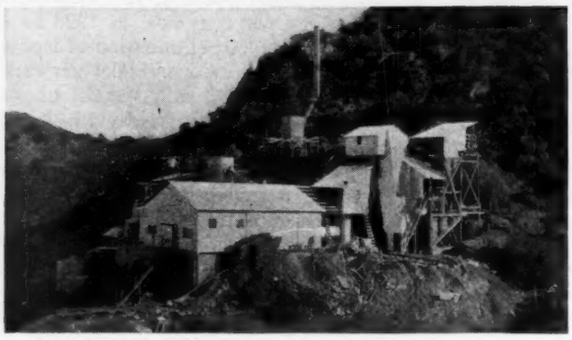
Fig. 2—Variations in mill operations 1923-1930

Gravity flow is made use of in the mill design wherever possible. Bucket elevators have been eliminated, dry material being handled by conveyor belts inclined upward where necessary to gain elevation. The mill consists of a series of stepped floors extending from the mill-bin floor to the concentrate delivery. Floors are of cement. All foundations are of concrete, and the structure is of steel.

Ninety-two pumps, of the Wilfley type, are used for handling pulps, being an important feature in the mill arrangement. Without an efficient pump the difficulties of mill design would have been great. Genter thickeners, which were developed at the plant, enable pulp dilution to be controlled. Tank thickeners conserve water. Vacuum pumps are contiguous to the filters. Reagent storage is at a central point to one side. Likewise a well-equipped repair shop included in the mill building serves both the fine-crushing and grinding sections. Motor drives are used throughout. Control switches are grouped in banks at convenient points. A boiler plant containing seven B. & W. boilers, each 318 hp., equipped with automatic stokers, supplies steam for the plant and also for three 1,500-kw. turbo-generators that supply auxiliary power. A separate laboratory, office building, and warehouse for supplies are near the main plant.

The concentrator installation is the epitome of technologic knowledge and practical experience, and reflects credit on the designers. Its operation, closely controlled as it is and approaching the limits of practical recoveries, is evidence of the ability and zeal of its operating staff, an energetic group of young engineers.

Recent Design of Quicksilver Plants



Plant of the Mercury Corporation of America, under construction near Santa Margarita, Calif.

Activities in quicksilver mining have been well sustained, despite the recent drop in the price of the metal, a drop that is not surprising in view of the present status of the non-ferrous metal industry. This activity has resulted in several changes in equipment and metallurgy of reduction plants.

C. N. Schuette

THREE types of furnaces are in common use—the old Scott furnace, the rotary kiln, and the Herreshoff furnace. The Scott furnace burns any type of fuel and requires neither power nor skilled labor. It has the disadvantage of not being suitable for intermittent operation, and operates only on a reasonably dry feed.

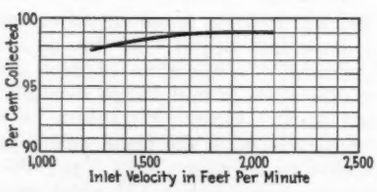


Fig. 1—Test results of Sirocco dust collector

An auxiliary dryer is therefore sometimes essential. Both the rotary kiln and the Herreshoff furnace are mechanical, power-driven furnaces, the operation of which, unlike that of the Scott furnace, produces a certain amount of flue dust. Any dust entering the gas stream must be removed before the mercury-laden furnace gases reach the condenser system. The Herreshoff furnace has the advantage of a higher fuel economy than the rotary kiln, and, unlike the kiln, no occasional shutdowns for relining, are necessary.

In plants operating mechanical furnaces, removal of dust from the furnace gases is accomplished by settling chambers, water sprays, Cottrell precipitators, or by various types of cyclone settlers. Recently, I have introduced the "Sirocco" dust collector, made by the American Blower Company, into quicksilver metallurgy and have found it to possess the advantages of low cost and efficient operation. An almost complete dust elimination achieved by this type of collector has made possible many other interesting features of design in plants recently built under my direction.

Efficient dust elimination has presented a problem ever since the year 1916, when the mechanical furnace was introduced. The Cottrell precipitator was used for the first time in connection with the original Herreshoff furnace at New Almaden. The first rotary kiln built at New Idria in 1917 was connected to large concrete dust chambers with water sprays in the flues beyond the chambers, to wash out any remaining dust. Later, cyclones of various types, single, in pairs, or in multiple units, were used at many plants, but difficulties of adjustment soon arose. The efficiency of removal varied with the amount, quality, and fineness of the dust. A draft, adjusted to give maximum dust collection in the cyclones, was not found suitable for efficient operation of the furnace. None of the systems in use removed all of the dust, and the problem of dust escaping into the condenser system was therefore not solved, conditions being only ameliorated.

Dust entering the condenser system is mixed with the finely divided particles of condensed mercury. When the amount of dust exceeds a certain minimum, separation of the mercury, by the usual method of hoeing on tables or by working the product in a type of "soot machine," becomes difficult. In some plants where the amount of dust entering the condensers is high, the condensed product is retorted a second time to recover the quicksilver. In others, part of the dust is removed by mechanical means and the remainder is treated by retorting. Either method adds to the cost of fuel and labor. If all the dust is returned to the kiln, the furnace capacity is reduced materially.

The Sirocco dust collector has been installed in new plants and has replaced older types of dust collectors in others. Wherever possible, samples of the dust to be collected are tested in laboratory machines. According to present experience, the indicated dust recovery of 98 to 99 per cent for inlet velocities ranging from 1,250 to 2,000 ft. per minute has been duplicated or exceeded in practice. The results of a preliminary test are shown in Fig. 1. Fig. 2 shows an installation of this collector on the rotary kiln at the Rinconada mine of the Mercury Corporation of America, at Santa Margarita, Calif.; and Fig. 3, one on the Herreshoff furnace of the Rainbow mine, at Terlingua, Tex.

The cyclone consists of a scroll-shaped top over a series of alternate cones and cylinders. Dust-laden gas passes downward with increasing velocity, the dust is thrown out, and the

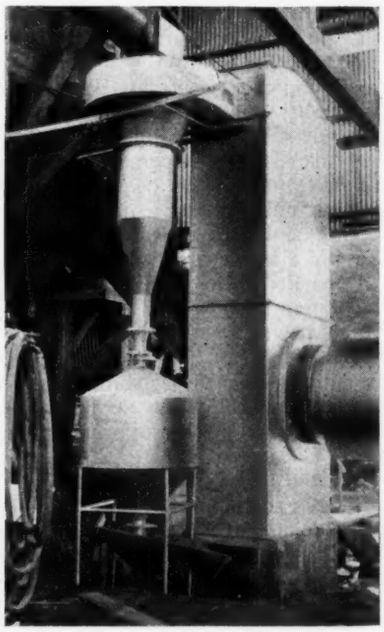


Fig. 2—Sirocco dust collector at the Rinconada mine. Note the narrow dust chamber.

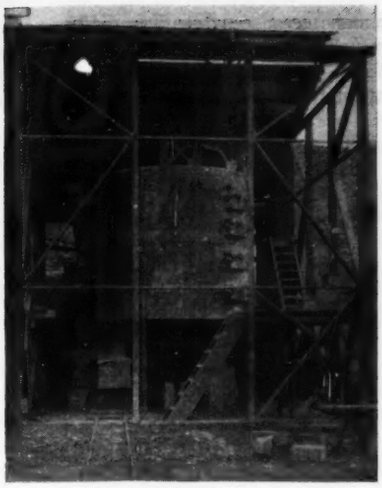


Fig. 3 — Four-hearth 10-ft. diameter Herreshoff furnace at the Rainbow mine. Dust collector at upper left.

gas ascends in the center to enter the scroll-shaped discharge head. The collector operates at high inlet velocity and high water gage. When the conditions of operation have been correctly calculated, practically all of the dust made is caught and eliminated from the mercury-laden gases.

In the installation described, the collector is placed as near the furnace as possible. When a Herreshoff furnace is used, it is attached directly to the condenser. Gas discharged from the collector is kept at a temperature several hundred degrees above the dew point of mercury for the conditions of operation. As the temperature drop in the collector is small, no insulation of the collectors has been needed in any of the five installations completed up to the present time. The cross-section of dust chamber shown in Fig. 2, to which the collector is connected, is so small that no settling takes place; it serves only as a flue for conveying the gases to the collector. Thus, dust removal is restricted to one point of discharge. The dust, caught in the collector receptacle, contains only a trace of mercury. As an example of this retention of metal, a unit treating 50 tons of ore per day and yielding an amount of dust equal to 2 per cent of the weight of ore produces one ton of dust, containing 2 or 3 lb. of mercury, daily—a small price to pay for the avoidance of dust re-treatment.

Condenser systems in present-day plants are practically all pipe systems, in contrast to the older chamber condensers. Monel-metal pipes, tried out some years ago, have nearly all been replaced, as their life was short and the cost high. Cast-iron pipes are used, but except under the most favorable conditions they pit and corrode badly on the inside after a short use. An internal crust decreases the cooling efficiency rapidly, although they may stand up a fairly long time. The first cost is also high.

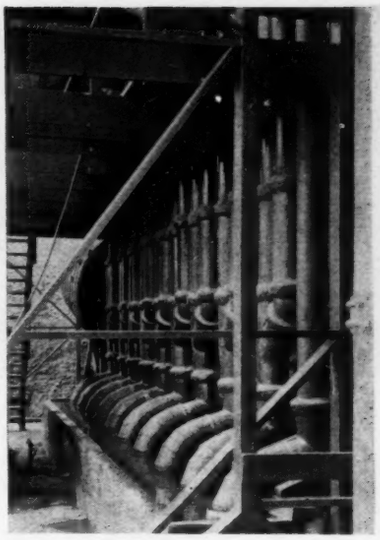


Fig. 6—Discharge spouts at the Rainbow plant, which is yielding mercury daily

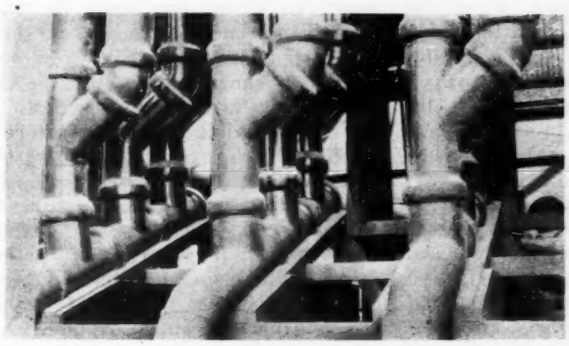


Fig. 5—Connection of the vertical columns with the inclined pipe. Note the sharp turn the gas must make.

Fig. 7 — Modified cast-iron pipe condenser, allowing daily "run-downs."

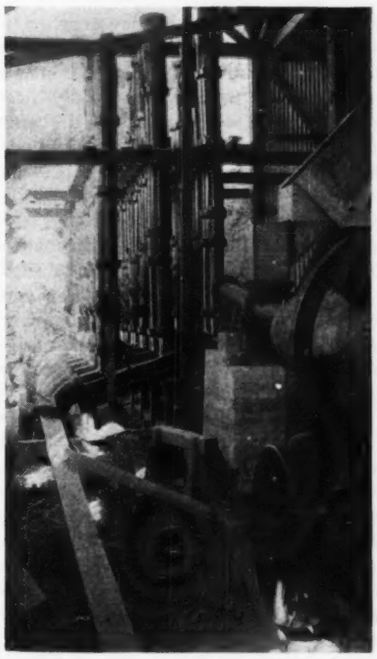


Fig. 4—End view of new type of tile condenser. A reciprocating kiln feeder is shown under the chute.

Salt-glazed tile pipe, in spite of the fact that cracks develop in use, is the most economical condenser material now employed. Washed down frequently, tile pipe remains as clean as

Table I—Screen Tests of Dust from Various Plants

Screen Analysis		Per Cent Passed					
Sieve No.	Opening Inches	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6
325	0.0017	28.45	78.1	89.26	99.4	97.76	86.5
200	0.0029	54.7	87.3	95.26	99.8	99.56	95.5
150	0.0041	69.7	92.7	98.40	100.0	99.75	96.5
100	0.0058	81.8	96.3	99.44	.....	99.83	97.5
60	0.0087	93.9	99.0	99.84	.....	99.90	98.7
40	0.0150	98.5	99.9	.....	.....	99.95	99.4
20	0.0340	.....	.....	.....	.....	.....	.....
Pounds per cubic foot	.....	67	62	60.3	40.35	71.5	32.00
Concentration, grains per cubic foot	.....	3	3	10	.....	3	10
Per cent recovery indicated by test at 2,000 ft./min	.....	99	98	99+	(a)	99	96

(a) Not tested. About 100 per cent recovery in practice.

Fig. 3—Tile pipe condensers, and redwood barrel at the Antelope Springs mine near Lovelock, Nev.

on the day of instaliation, whereas metal pipe gradually deteriorates. Dust, passing into a condenser system, forms a crust on the walls, thus lowering the cooling power and impeding the draft. Tile pipe of small diameter, 12 in., often fills up with dust in a week. The almost perfect elimination of dust accomplished by the Sirocco collector permits the use of pipes of smaller diameters, such as 8- and 10-in. pipe, where diameters up to 3 ft. were formerly specified. This offers many advantages. The gas velocity in the smaller pipes is higher, a factor materially increasing their cooling power. They are structurally stronger and offer better resistance to variations in temperature; they are easier to ship and to handle during erection; and the cost per square foot of cooling surface is lower than that of large-diameter pipe.

A new type condenser system made from tile pipe is illustrated in Figs. 4 and 5. Eight-inch salt-glazed pipe is used in three rows of columns, about 10 ft. high. The gases flow up one column of the row and down the next. Connections at the top are made with standard T-branches. At the bottom, 45-deg. branches and a curve are used to make the connection, shown in Fig. 5. Sharp changes in the direction of the gas flow result, and the baffling action, which throws down the condensed mercury, is excellent. Instead of leading into a concrete chamber, the condenser pipes are set on a sloping branch pipe, at the lowest joint of which an elbow dips below the water level of a concrete trough, as shown in Fig. 6. This avoids use of the leaky concrete chambers formerly necessary, and provides a positive water seal.

Vertical and inclined pipes are each closed with tile covers equipped with 1-in. holes, closed with cork stoppers. A two-gallon pan of glazed stoneware is submerged. Each day the clean-up man inserts the nozzle of a hose through the 1-in. holes in the pipe covers and flushes down first the vertical and then the inclined pipes, washing the finely divided mercury into the pans. These are then removed, the water is decanted, and the mercury poured out on the soot table. As soon as it is dry, the mercury runs off, leaving a few handfuls of residue, which, if no lime is used, is thrown back into the furnace. This daily run-down takes only a few hours of one man's time, and the greater part of the product can be bottled daily. The remainder is recovered on monthly clean-ups, when all covers are removed and a thorough cleaning and inspection is made of the entire condenser system. This method of operation has particular advantages for the small producer, who must sell his quicksilver as fast as he produces it, to meet payroll and running expenses.

Fig. 7 shows the application of the system to a series of cast-iron condensers that formerly were difficult to clean. One of the stoneware receptacles can be seen at the left in the picture.

Following the tile pipes is a four-compartment redwood barrel, shown in Fig. 8, usually placed ahead of the flue leading to the stack. This acts as a safety measure in the event of improper furnace operation. As a rule, little mercury is recovered at this point provided the tile-pipe unit is of correct proportions.

A condensing system should be under draft, a rule that is not always followed. Leaks may occur, resulting from faulty connections or other defects. Unless the system is under draft, fumes carrying mercury will escape, causing loss and presenting the danger of mercury poisoning.

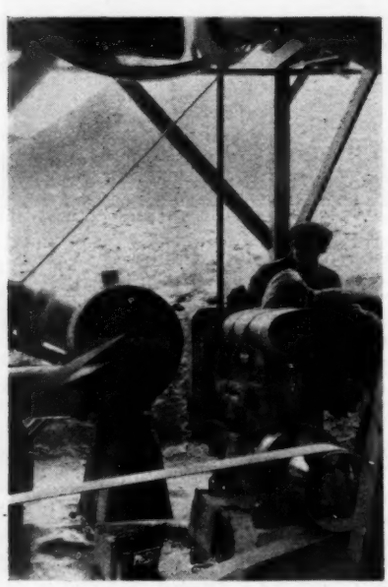


Fig. 9—Acid-proof exhauster at the Rainbow mine, in Texas. The engine in the foreground furnishes power for the plant.

Draft is provided by an acid-proof exhauster shown in Fig. 9, manufactured by California Pattern Works, of San Francisco. It has a slotted rotor hub of cast iron and wedge-seated fan blades of redwood. The scroll is made of redwood, with a removable top for cleaning, and drain bottom.

Draft is checked at various points along the condenser system with U-tube gages. The resistance of the pipes is generally not more than $\frac{1}{8}$ to $\frac{1}{4}$ in. of water. Any greater difference between the gages at the entrance and the exit of the system indicates a leak that should be corrected.

A condenser system under draft can be cleaned without any interruptions in the furnace operation and without danger to the crew. With the new form of condenser the great part of the production can be bottled daily; no dust has to be re-treated when a Sirocco collector or a mechanical furnace system is used.

■

New Process Employs Chloridizing Roast

COMMERCIAL application of the new process for treating oxidized and semi-oxidized copper ores is now being made by Southern Rhodesia Base Metals at its Alaska mine near Sinoia, Southern Rhodesia, this mine possessing large tonnages of such ore. The process was developed by Minerals Separation.

A Dodge crusher reduces the ore to $\frac{3}{4}$ in., after which it is dry-crushed in rolls in closed circuit with a 10-mesh vibrating screen. It is then elevated to a Herreshoff furnace, externally fired. By the time the ore reaches the sixth hearth of this furnace it has attained a temperature of about 650 deg. C. On the sixth hearth a mixture of carbon (coal, charcoal, or coke) and salt (sodium chloride) is added, the carbon amounting to 2 per cent of the weight of the ore and the salt to $\frac{1}{2}$ per cent. The two lower hearths, Nos. 6 and 7, are muffled to exclude furnace gases and to maintain the temperature while reduction to metallic copper takes place.

The ore is discharged through a pipe, falling into a slowly revolving cylinder, sealed to exclude air and cooled by external water sprays. Water is then added, together with suitable reagents, and the pulp elevated by a sand pump to a sub-aëration flotation machine. In this a high-grade concentrate is made, containing only about 5 per cent of the gangue and some of the carbon which was in excess of that required for the reducing roast. After filtering, it is bagged for shipment to the smelter.

Magnetite in Copper Matte

PRESENCE of magnetite causes copper losses in the slag from the smelting of matte. This study of the results of experiments on the distribution of the magnetite should prove of help in the solution of the problem, as well as demonstrating the value of improved chemical and microscopic methods of analysis.

F. G. Hawley

COPPER MATTE is generally considered to be a homogeneous mixture or solution of copper and iron sulphides, together with smaller amounts of sulphides of other metals, such as lead, zinc, and antimony. Nearly all metallurgists agree that in ordinary copper matte the copper exists as cuprous sulphide (Cu_2S), and the iron sulphide approximates more or less closely to ferrous sulphide (FeS), though usually the sulphur found by assay is considerably less than the theoretical amount needed to form cuprous and ferrous sulphide and to combine with the smaller amounts of the other metals. So some metallurgists have assumed that part of the iron exists as a solution of metallic iron, dissolved in the sulphide. Others postulate a complex iron sulphide containing less sulphur than is required for ferrous sulphide.

For many years metallurgical technicians have known that copper matte, especially when low grade, frequently contains magnetite, but comparatively few data have been published on the subject, and the fact is not generally realized that it is universally present and in considerable amounts. Inasmuch as magnetite is usually considered detrimental to reverberatory smelting, the manner and cause of its occurrence are important. Lack of a simple and accurate method for its determination has prevented the securing of more definite knowledge on this subject.

A few laboratories have estimated magnetite by means of a magnetic balance. The apparatus is first calibrated by measuring the magnetic susceptibility of known mixtures of magnetite and some inert substance. By making similar measurements on matte, the magnetite can be calculated easily. Such a determination can be quickly made and may be accurate enough for many purposes. Its accuracy may be questioned, however, for the magnetic susceptibility of the magnetite in different mattes may vary. The susceptibilities of different natural magnetites vary decidedly, in fact, and so do those of different kinds of iron and steel that contain various small amounts of other elements. Physical differences, as those caused by heat-treatment, may have considerable effect; and this is also true of magnetite occurring in matte. Most mattes contain small amounts of many elements, with some of which the magnetite might form a solid solution or a molecular replacement.

I have not tested many samples of magnetite obtained from matte for unusual elements, but on one occasion found in two carefully purified samples, 0.42 and 0.35 per cent of molybdenum. Recently, on a sample of magnetite separated from matte from another smelter, I found 1.2 per cent of zinc and a little nickel.

Since publication of my article "Determination of Magnetite in Matte and Slags" (*E. & M. J.*, Aug. 23, 1919), the process, especially the modification for matte, has been improved. Results by this method are now quite reliable and the error should not be greater than one part in twenty parts or more.

At the International smelter occasional tests of matte for magnetite have shown its presence in considerable quantities. Ordinarily, the copper and iron content remain fairly constant over long periods of time. Occasionally, however, they fluctuate to some extent. When the grade of matte remained about the same, the magnetite was surprisingly constant; but during the infrequent intervals when the grade of the matte did vary, the amount of magnetite also varied, and in almost direct ratio to the iron present and consequently in inverse ratio to the copper.

About a year ago, experimental work was started here to investigate the conditions governing the distribution of magnetite in matte. Samples of varying grades from this smelter, and also samples of high, medium, and low grades from eight other smelters of the Southwest, were obtained. In all, 35 samples were examined, 29 of them being from reverberatories and 6 from blast furnaces. The copper and iron content varied widely, one extreme of the reverberatory series containing 47.12 per cent of copper and 24.8 per cent of iron, and the other extreme had 21.35 per cent of copper and 44.1 per cent of iron. Careful and nearly complete chemical analyses of these mattes were made, including the determination of magnetite. Qualitative tests showed little arsenic and antimony, so these determinations were not included. Some microscopical examinations were made, in part checked by the Bureau of Mines at Salt Lake City. Small portions of the samples were sent to the Phelps Dodge laboratory at Douglas, where Mr. L. L. McDaniel made determinations of magnetite by means of their magnetic balance. On many of them, the results checked ours fairly well, but on others very poorly. In general, their results confirmed the rule that magnetite increases with the iron.

Tabulated chemical analyses resulting from the work are given in the following table, in which the ratio of magnetite to sulphide iron is given as *K*, i.e.

$$\frac{\text{Magnetite}}{\text{Sulphide iron}} = K.$$

No mattes extremely high in copper were available for analysis, though a partial analysis of three samples of white metal gave the following results:

Per Cent		
Cu	Total Fe	Magnetite
76.38	2.2	0.85
79.08	0.7	0.19
79.60	0.65	0.19

The analyses of the blast-furnace mattes are somewhat erratic, and as two smelters only are represented, and the two series are dissimilar, definite conclusions cannot well be drawn from them. On the other hand, most of the findings on the 35 reverberatory samples are more consistent and seem to justify certain conclusions. The following discussion, unless otherwise stated, is confined to the reverberatory mattes.

All samples contained considerable magnetite, in amount roughly proportional to the iron. Possibly the relationship should be considered as between the magnetite and the ferrous sulphide, or between the magnetite and the iron of the ferrous sulphide. For practical purposes, which factor is used makes little difference. In determining the ratios, the sulphide iron was calculated by subtracting the iron in magnetite from the total iron. No correction was made for a very little iron in slag contaminations.

Matte Analyses

	Cu	Fe	Zn	Pb	S	O	"Insol."	Total	Fe_3O_4	K
Reverberatory Smelter "A"	47.12	24.8	1.10	Nil	24.8	1.6	0.3	99.72	5.5	0.26
	44.47	27.2	.70	Nil	25.1	1.82	0.35	99.64	6.6	0.29
	42.21	28.8	1.10	Nil	25.1	2.00	0.2	99.41	7.3	0.31
	36.62	33.1	.95	Nil	25.2	2.5	0.25	99.62	9.0	0.37
	29.20	39.0	1.10	Nil	25.5	3.3	0.45	98.55	11.7	0.38
Reverberatory Smelter "B"	44.84	26.1	1.10	Tr	21.6	2.48	3.00	99.12	9.0	0.47
	36.17	33.8	1.00	Tr	24.9	2.84	0.60	99.31	10.3	0.39
	28.00	40.5	1.05	Tr	24.8	3.42	1.40	99.17	12.4	0.39
Reverberatory Smelter "C"	38.19	31.7	1.20	0.67	24.2	3.06	0.45	99.47	11.1	0.47
	29.96	37.6	1.40	0.53	25.2	3.48	0.55	98.72	12.6	0.44
	24.82	41.8	1.50	0.60	26.2	3.75	0.55	99.22	13.6	0.43
Reverberatory Smelter "D"	31.08	36.5	2.50	0.15	24.3	3.95	0.90	99.48	14.3	0.55
	26.17	40.1	3.20	0.17	24.6	4.28	0.70	99.22	15.5	0.54
	21.35	44.1	3.35	Tr	24.0	3.92	1.05	98.77	17.8	0.57
Reverberatory Smelter "E"	39.39	29.3	2.50	0.25	24.0	2.76	1.20	99.40	10.0	0.45
	30.33	36.3	2.85	0.22	23.9	3.45	1.70	98.75	12.5	0.46
	25.01	40.6	3.05	0.22	24.3	4.28	1.30	98.76	15.5	0.53
	21.79	42.4	3.45	0.22	25.2	4.28	1.40	98.74	15.5	0.50
Reverberatory Smelter "F"	43.58	26.8	1.25	0.36	24.8	1.90	0.40	99.09	6.9	0.32
	39.46	30.5	1.20	0.33	25.1	2.37	0.35	99.31	8.6	0.36
	34.98	33.8	1.50	0.39	25.6	2.73	0.35	99.35	9.9	0.37
Reverberatory Smelter "G"	38.17	31.9	0.6	Tr	26.3	1.96	0.35	99.28	7.1	0.26
	35.86	33.8	0.6	Tr	26.5	2.26	0.35	99.37	8.2	0.29
	29.36	39.1	0.65	Tr	27.0	2.79	0.45	99.35	10.1	0.32
Reverberatory Smelter "H"	34.90	34.3	1.30	0.15	25.5	2.35	0.75	98.95	8.5	0.30
	32.70	36.2	1.35	0.10	25.5	2.84	0.70	99.37	10.3	0.36
	28.81	39.4	1.20	0.10	26.5	2.84	0.65	99.50	10.3	0.32
Reverberatory Smelter "I"	41.51	26.9	1.55	1.40	26.0	1.27	0.45	99.08	4.6	0.20
	40.33	27.1	1.55	2.65	25.9	1.87	0.30	99.70	6.8	0.31
Blast Furnace "D"	21.31	44.7	2.85	0.13	25.2	3.26	0.55	98.00	11.8	0.33
	17.71	47.1	3.60	0.11	25.3	3.26	0.75	97.83	11.8	0.31
	14.64	49.0	4.00	0.11	24.3	3.48	1.00	96.53	12.6	0.32
Blast Furnace "E"	29.62	36.7	3.05	0.13	26.0	1.10	1.10	97.70	4.0	0.12
	28.58	37.0	3.70	Tr	26.8	1.16	0.70	97.94	4.2	0.12
	26.03	39.3	3.65	Tr	27.5	1.32	0.55	98.35	4 8	0.13

Comparison of the ratios of magnetite to sulphide iron shows that the average ratios from the different smelters differ more than do those on individual samples from the same smelter. Thus, the average ratio of magnetite to sulphide iron at one smelter (Reverb. "D") is 0.55, whereas at another (Reverb. "G") it is 0.29. At the International smelter the ratio for individual samples, where the grade varies, runs from 0.26 to 0.38. On ordinary medium-grade matte it stays nearly constant for months at a time at 0.31 to 0.34.

The average ratios of magnetite to sulphide iron for the two blast furnaces differ widely, one being 0.32 and the other only 0.12, the latter being a much lower ratio than was found in any reverberatory.

Referring again to mattes made at the same smelter and of different grades, an examination of the analysis shows that with many of the series an increase of iron is followed by an even greater proportional increase in magnetite. This is especially noticeable in our matte, where the ratios increase quite uniformly as the following figures show:

Fe in FeS Per Cent	Magnetite Per Cent	Ratio
20.8	5.5	0.26
22.4	6.6	0.29
23.5	7.3	0.31
24.6	9.0	0.36
30.5	11.7	0.38

That the figures are not as significant as they appear is possible, for at this plant very high- and very low-grade matte is made only occasionally and then only under such unusual conditions that we have few opportunities to check the figures.

The mattes from four of the nine reverberatory smelters appear to follow the general rule of increasing ratios; three are rather constant; and two of the series show decreasing ratios. One of these decreases only slightly and the other is represented by only two samples and so is not very significant. One of the series of the two blast furnaces shows nearly constant ratios. In the other the ratios increase.

Undoubtedly the conditions under which the charge is smelted will, to some extent, affect the amount of magnetite. Some of these conditions probably are the temperature in the furnace; the speed of smelting, and amount of reducing agents present.

Before magnetite will combine with the slag chemically it must be reduced to ferrous oxide. Experimental work done at the Bureau of Mines station at Tucson (See "The Reaction Between Magnetite and Ferrous Sulphide," by F. S. Wartman and G. L. Oldright) indicates that, under the conditions of most reverberatory smelting, the reduction would be rather slow. Our experience seems to confirm this. Probably considerable reduction takes place at or near the surface of the charge where the temperature is highest, but if the magnetite has entered the matte where the temperature is lower it is very slow.

Data covering all of the conditions under which the mattes were made were desirable, but to obtain them proved to be impracticable. Most of the samples represented matte that had been smelted several months previously.

Occurrence and Physical Properties

If matte is ground to 80 or 100 mesh and treated with a strong solution of sodium chlorate in nitric acid, the sulphide will be dissolved quickly, but if the treatment is not too hot or too prolonged, the magnetite is attacked only slightly. After removing the acid, washing and drying, the magnetite can be examined with a miscroscope or otherwise. Magnetite so obtained showed plainly that practically all of it consists of minute crystalline grains. Mr. R. E. Head, after examining three samples, describes it as being composed of crystallites of dendritic form, frequently evolving a skeleton structure that sometimes inclose a little slag. The size varies from 0.1 mm. to 0.001 mm., and even smaller. He found a very little ferric oxide in each sample and roughly estimated the amount at about 1 per cent of the impure magnetite. This small amount might have come from the iron ladle used in sampling.

The extreme smallness of the magnetite crystals may be better understood when the fact is set forth that their average size is about one-tenth of that of the opening of a 150-mesh screen. Much of this material is too fine to settle readily in water. Apparently, only little if any coarse magnetite was present in the original matte. Otherwise, in crushing to 100 mesh such crystals would have been broken into rough pieces. On dissolving the sulphide, this magnetite should remain as relatively large pieces and show little crystalline structure. Very few such pieces were ever found.

Any sample taken from the same run of matte will contain nearly the same amount of magnetite, proving its even distribution. Another fact that shows the extreme smallness and thorough distribution of the particles is that no magnetic separation of magnetite from the matte is possible.

The test is best made under water and with an electromagnet of variable strength. Undoubtedly the particles of magnetite are so minute that they are not liberated from the small grains of matte and so make the whole magnetic. Pure ferrous sulphide is not magnetic to any extent, but the commercial article is, as it usually contains 7 to 10 per cent of magnetite.

Effect of Specific Gravity

High-copper low-iron matte is heavier than magnetite, and low-copper high-iron matte is lighter; and thus magnetite should seemingly float on a low-iron matte and sink into one high in iron. This has been given as the reason why matte with high iron contains much magetite. It may be one reason, but I think it is not the principal one. If magnetite can sink into a high-iron matte, it should be able to sink through it and onto the bottom, so that nearly as much would leave the matte as enters it. On the other hand, if the specific gravity of the matte is high enough to cause the magnetite to float, none should enter the matte. According to this, a curve representing the magnetite in different grades of matte should show a fairly sharp break at, or near, the point where the specific gravity of the matte equals that of the magnetite; at which point the matte should be able to hold the maximum amount of magnetite. This is contrary to our analysis, which shows a uniform increase.

The question seems much more com-

plicated than the foregoing conclusions might indicate. Other factors, such as bubbles of sulphur dioxide generated by the reaction between the magnetite and ferrous sulphide, may tend to keep the particles of magnetite in suspension and may influence the action. To have a mushy blanket of magnetite, which probably contains numerous tiny bubbles or gas pockets of sulphur dioxide, floating between the slag and matte, and at the same time a solid mass of magnetite below the matte, is a common occurrence.

A theory—it is not entirely original—that may account for the facts observed, is that the magnetite is soluble in the molten matte, the solubility increasing with the amount of ferrous sulphide present. To say that it is soluble in ferrous sulphide but not in cuprous sulphide may be more accurate. The solubility probably increases somewhat with a rise in temperature. The amount of magnetite eliminated from the furnace is usually somewhat less than that contained in the charge. The excess is stored up as concretions on the bottom and sides. If for a time less enters the charge than is eliminated, the matte will dissolve some of the excess from the bottom and so keep itself saturated. This would account for similar mattes always containing nearly the same amount of magnetite. The magnetite found in the bottom of reverberatories usually is made up of aggregates of comparatively large crystals. Probably the cooler bottom causes the matte at that point to be supersaturated, so that some magnetite crystallizes out. The crystals are large because the growth is slow and even.

When matte is tapped and suddenly solidifies the magnetite quickly separates. The sudden cooling would account for the very minute size of the crystals. There would be no time for segregation and hence the perfect dissemination. Some of the magnetite occurring in the liquid matte may be in the form of small crystals held in suspension, but I think the amount is much less than that held in solution.

Comments on the Analytical Data

The summation of the chemical analysis of the matte is in general satisfactory, usually totaling over 99 per cent. Where it is less than this the "Insoluble" is frequently found to be high, indicating some contamination of slag. This would introduce small amounts of alumina, lime, magnesia, and alkalies, which were not determined. This slag contamination usually appears to be more common in matte high in iron. No simple method for the direct determination of oxygen in matte exists, but to calculate that present in the magnetite is easy, and this was done.

Most of the analyses showing high magnetite have a summation lower than those with a low magnetite content. This may be used as an argument against any claim that the magnetite determination is too high, for the less magnetite assumed to be present, the less oxygen would be assigned to it, and the summation would be still lower. By leaving the oxygen of the magnetite out of the analysis the sum of the other elements for mattes high in copper would be about 98 per cent, and for low-copper high-iron mattes it would be only 94 to 95 per cent.

If iron equivalent to that of the magnetite is subtracted from the total iron, we get sulphide iron. If we now calculate the sulphur necessary to form ferrous sulphide with this, and add it to the amount necessary to form cuprous sulphide and that needed to combine with any lesser metal present, the result will check very closely with the sulphur found by assay. In other words, the sulphide iron corresponds closely to the theoretical ferrous sulphide.

One chemist has suggested that during the magnetite determination the strong oxidizing agent used to remove sulphides might form magnetite from the iron sulphide. This did not seem reasonable, but, to make certain, some tests were made with chemically pure ferrous sulphide and also with ferrous sulphide combined with cuprous sulphide. The experiment was repeated several times in different ways, but no magnetite could be made.

The chemical formula for magnetite is Fe_3O_4. This is considered as being composed of one molecule of ferrous oxide combined with one of ferric oxide. Dana gives the composition as ferrous oxide 31 per cent and ferric oxide 69 per cent. Following are the analyses of magnetite separated from two typical samples of our matte:

Analyses of Magnetite

	Per Cent	
	No. 1	No. 2
FeO	33.5	32.7
Fe_2O_3	62.8	63.1
Cu	0.15	0.08
Mo	0.42	0.35
S	0.28	0.24
"Insoluble."	1.3	2.9

These percentages are about what one would expect from slightly impure magnetite. Obviously, the ferrous oxide is somewhat higher than theoretical. One reason is the presence of a little ferrous oxide from the small amount of slag in the sample; another is that in the determination of ferrous iron a little ferric iron is reduced by the trace of sulphide present.

Method Used

The method used for separating the sulphide from the magnetite is as follows:

Dissolve 30 grams of sodium chlorate (not potassium chlorate) in 50 c.c. of hot water. Cool and add 50 c.c. of nitric acid; cool again and add 30 c.c. of 85 per cent sulphuric acid and cool to room temperature or lower.

The sodium chlorate must be entirely in solution when the acid is added. We have never had an explosion with this mixture, but small explosions might occur if it were improperly made or used. Weigh 0.5 gram of the matte into a 600-c.c. beaker and put in eight to ten drops of water. Shake to spread the matte over the bottom of the beaker; then put it in a cooling trough with cold water and while gently shaking add 25 c.c. of the chlorate mixture. In a few minutes, when the action has moderated, heat the beaker nearly to boiling, then add 50 c.c. of cold water and immediately filter and wash. A one-gram sample can be used if five to ten grams of ice in small pieces is added before the chlorate mixture. The magnetite can be determined by weighing as such, and then making a correction for the impurities present, or by igniting and weighing as ferric oxide and deducting impurities as before, or by determining the ferric iron volumetrically and calculating the magnetite.

After this article was partly written, my attention was called to the fact that on Nov. 16, 1895, *Engineering and Mining Journal* published an article by Edward Keller on the "Magnetic Oxide of Iron in Copper Matte." Keller calls attention to the existence of magnetite in matte and shows that the lower grade of matte contains the most magnetite. He gives some quantitative data. Many of our conclusions are similar, but in some we differ. His method of separating sulphide from magnetite somewhat resembles that given here, but the solvent used was not nearly so efficient, requiring more time and higher temperatures, which would cause a somewhat greater solubility loss of magnetite.

The Witwatersrand

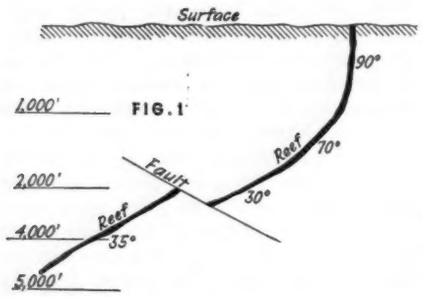
Evolution of Mining Practices

Especially in the Central Rand

Bernard Beringer

Morococha, Peru

THE pioneers of the Rand consisted of British and American engineers, supported by miners from Great Britain. Those days were characterized by a dearth of technical men in the mining world; and, although the Rand was able to boast of good organizers and capable technicians, a detail of underground work such as stoping practice was left to the mine captains of Cornwall and the North of England. They were the leading miners of their day, but, unable to work from first principles, they were prone to favor practices to which they were used rather than to embark on experiments.



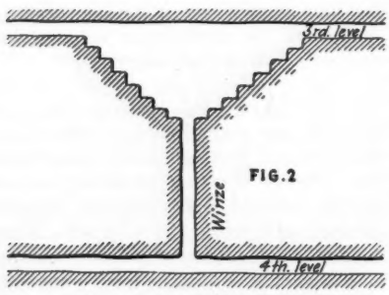


Fig. 1—Transverse section through the reef; Fig. 2—Underhand method of stoping

Up almost to the time of the Boer War the engineers of the Rand had well-nigh insuperable difficulties to face in the question of transport, the railhead being at Kimberley. From there ox wagons brought to the mines the heavy machinery needed. Later, when the railroad was built to Johannesburg, the Boer Republics placed artificial restrictions which tended to cramp the growth of what was becoming the largest gold-mining area in the world. However, those early consulting engineers and managers won through. By adopting the latest methods known, they developed sound mining practice. For example, as the technique of electrical practice improved, they discarded Cornish pumps, replaced combined shafts by stage winding, and gradually began to lead the world in efficient layout of the mines and a sound development policy. Sampling was brought to a fine art soon after the adoption of the cyanide process, which made mining of low-grade ore possible.

The one great failure of the Rand, which lasted up to nearly twenty years ago, was that little or no attempt was made to study ventilation and the avoidance of silicosis, which caused a colossal annual death rate from phthisis among the whites and the black miners. The Rand, like the rest of the world, failed also to revolutionize stoping and drilling practice until recent years, leaving the so-called practical man to continue in his conservatism.

The great improvement made in the drilling and blasting of holes dated from 1922. Improvement in stoping methods commenced, fortunately, at an earlier though belated date. Fig. 1 depicts a transverse section through the reef of what will be called the "X" mine. It will be noted that the orebody in this particular mine varied in dip from vertical to 30 deg. The question of dip is the most important in deciding a method of stoping tabular deposits; hence the "X" mine can provide a more embracing history of stoping than mines with a more or less consistent dip. In width, the orebody varied from 1 ft. to 8 ft. From the surface to 1,500 ft. will be considered first, because the problem is concerned with dips of between vertical and 45 deg., and to which the same methods of stoping are applicable.

What is known as the underhand stoping, or Cornish, method, shown in Fig. 2, was most favored in the initial stage of the "X" mine. The stope, worked from a raise or a winze, is opened up from the top downward. The disadvantages are obvious: (1) Gravity, which at these dips could, at no cost at all, transport the broken ore to the trammers in the level below, is interfered with, and the ore has to be shoveled down the stope faces. (2) Misfires are possible because many of the holes are wet, and the misfires are not readily seen. (3) A fall of hanging may roll down and kill the drillers. (4) Ventilation is poor, because, when the winze becomes full of broken ore, there is no through current of air. This unfortunate condition continues until the stope has progressed back from the original winze. (5) Inconvenience is caused by accumulated ore at the points.

The first advance was to improve the ventilation so as to avoid delay in reentering the stopes after blasting and also to lessen the effect of disadvantage (5). The hanging support was also strengthened. In this way was evolved

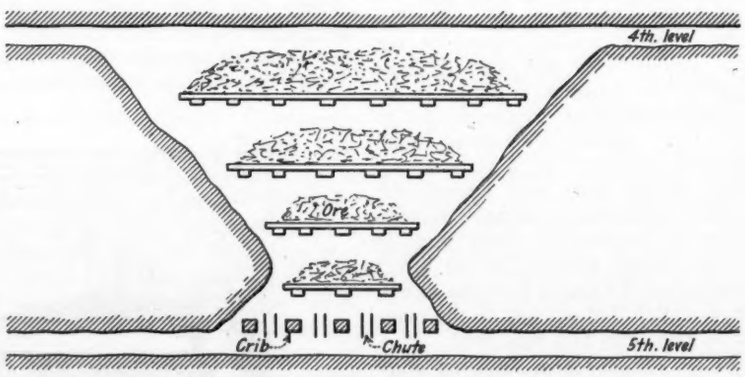


Fig. 3—Underhand stoping onto stulls (longitudinal section in plane of reef)

underhand stoping onto stulls (Rand method), shown in Fig. 3. The face is carried at 45 deg. from the horizontal, as before. Stulls are built across the stope at 20-ft. intervals, and broken ore is allowed to accumulate on these, thus supporting a larger area of hanging wall and affording overhead cover for the drillers. Stulls in heavy ground may be strengthened by filled cribs, which

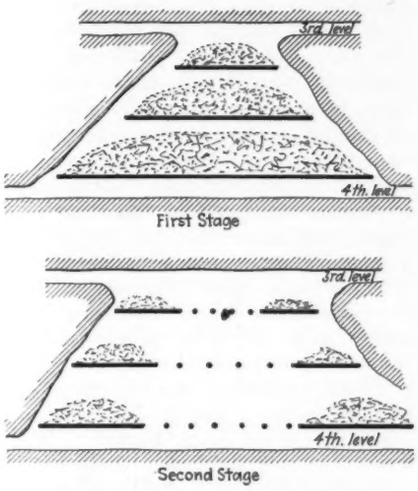


Fig. 4—Overhand stoping onto stulls (longitudinal section in planes of reef)

do not give way as do stull props. In the sketch the toes of the stope are shown as beaten back, to facilitate the drawing of the ore after it has been shoveled down the stope faces.

Accessibility and ventilation were improved, safety was increased to a certain extent, and the drawing of ore accelerated. But the great improvement in stoping did not come until underhand stoping was abolished for these dips of 45 deg. to vertical, and replaced by overhand stoping, which has such advantages over underhand stoping for steep dips that one wonders why, in some mines, the method has not yet been adopted. First, overhand stoping is more economical. No shoveling by hand or by machine is required. Second, the stope faces are left clean by the blast and the hole sockets (bootlegs) readily examined. Third, safety is increased because a fall of hanging can hurt only the man beneath, and a fall of hanging back from the face can hurt nobody. Fourth, the air current is directed up either stope face, insuring perfect ventilation.

Fig. 4, depicting overhead stoping onto stulls, is applicable to deposits up to 12 ft. in width and which dip at 45 deg. or over. The stope face is carried at a natural angle, having the advantage of being drilled by "down" holes, which are easier drilled than uppers, the weight of the hammer or of the machine helping the work. The stulls shown, placed about 20 ft. above each other, should be kept well loaded with broken ore, so as to afford standing room for the drillers and to support a larger area of hanging. Loaded cribs may be used where necessary, to strengthen the stulls. Overhead cover is provided by either of the stope faces, which are pinched free of loose rock, from the top down. As the stope faces advance the stulls are drawn inward and the broken ore that has been allowed to accumulate gravitates to the drive chutes. In this manner no ore need lie in the stope for more than a month. No argument about sunken capital need be considered.

Another form of overhand stoping, shrinkage stoping, with waste-filled compartments alongside the ore compartments (Fig. 5), would perhaps have been adopted were the stopes at a greater depth. The advantages of waste filling were not realized when the mines were shallow, but modern knowledge indicates the desirability of filling all stopes so as to lessen the worries of the engineers who will later have to stope the same orebodies at greater depths. Thus, although shrinkage stoping was not necessary at that time in the "X" mine, the overhand stopes should have been filled before abandonment.

For the deeper-level stopes of this mine, the dip of the orebody being about 35 deg., a modified form of underhand stoping was found most applicable. Practice proved that this method of stoping is as desirable at dips from 30 to 40 deg. in the mining of tabular deposits as for dips steeper than 45 deg. Fig. 6 depicts such a stope. The stope faces are carried as steeply as may be practicable, so as to interfere as little as possible with the gravitation of the ore, and yet to prevent the drillers from sliding down the stope. The footwall supports the men and the face checks them. No method of stoping at these dips permits the ore to gravitate without assistance. Therefore, overhand stoping would not be justified, and also much expense and delay would be caused by erecting temporary timber supports each shift, to enable the machine men to drill.

In the example shown, the hanging wall is supported by cribs made of 8 ft. eucalyptus logs and filled with waste or ore, staggered so as to afford overhead protection. With dips of 35 to 40 deg., overhead cover is augmented by a few loaded stulls, but at lesser dips the stulls are not necessary.

When the stope is worked out, those cribs which contain ore are emptied, and the filling of the whole area with waste rock is commenced. As the stope is being filled, the timber of the cribs is reclaimed and used elsewhere.

An impression exists that Rand stoping practice has become so stereotyped that it is not often applicable to the mining of tabular deposits in other parts of the world. This, I think, is erroneous, because Rand engineers have to stope at dips of from vertical to nearly flat, and at widths of from 1 ft. to 40 ft. They are thus familiar with tackling any mining problem that does not come under the heading of "Massive Deposits," for the mining of which type we have to look for experienced advice to the American and the Australian engineer.

The foregoing paragraphs have demonstrated the manner in which stoping methods improved, but the first real advance in actual rock breaking did not come until the reciprocating machine gave place to the hammer drill in stoping, about 1913-15. Also, by that time, nearly every mine had developed a sound ventilation scheme. Today the Rand mines are possibly the best ventilated in the world. The period noted was the time for serious research work on rock breaking and underground organization. Unfortunately, however, the War deprived the mines of many of their best officials and miners, and thus the managers had to make the most with a diminished staff and an inferior white labor list. Greatly to their credit were they able to keep production going. Thus, from 1915 to 1922 no great improvement was registered in the methods of breaking ground.

In 1921 (prior to the great strike), 26,900,000 tons of ore was mined at a cost of 25s. 8d. per ton, producing 8,128,000 oz. gold. Compare this with 1929, when 37,000,000 tons of ore was mined, at a cost of 19s. 7d. per ton, producing 10,414,000 oz. gold. Working costs were reduced by 6s. 1d. per ton, increasing dividends and adding many years to the life of the mines by bringing into the ore reserves many millions of tons that was previously unpayable. The grade stoped in 1929 was 10 per cent lower than in 1921. This is creditable, though at first sight it may not appear so. The total profit was about £7,500,000 more, which is what counts. Thus, then, in the period under discussion, working costs per ton were reduced by about 25 per cent, ac-

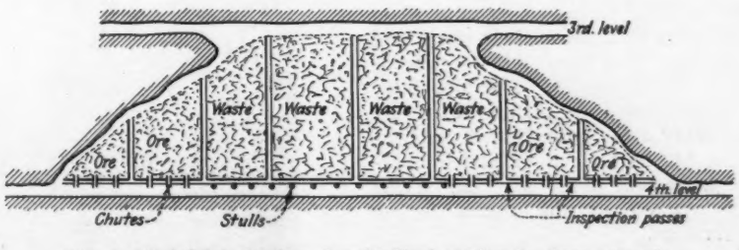


Fig. 5—Shrinkage stoping (longitudinal section in plane of reef)

companied by a large increase in profits. Reduction of working costs and a little increase in profit would have meant that improved underground efficiency was being nullified by a poor overhead policy; but, fortunately, efficiency and sound policy went hand in hand.

As to how the costs per ton were reduced from 25s. 8d. to 19s. 7d., the attributing causes were as follows:

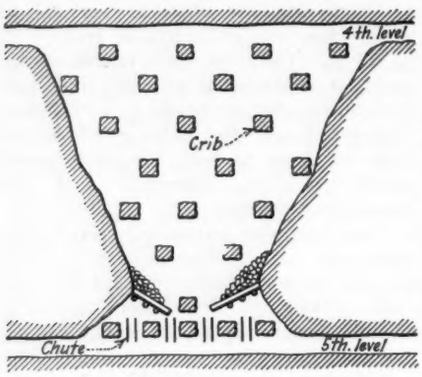


Fig. 6—Modified form of underhand stoping (longitudinal section in plane of reef)

Increased output per man, from (1) the establishment of discipline whereby, without altering the hours of labor, 20 per cent more hours were worked; (2) abolishing the contract system; (3) bringing in the system of payment by results; (4) granting a bonus for efficiency; (5) the drilling of only one hole per bench in stoping, and that pointed by an appliance; (6) reducing the size of drill-steel to $\frac{7}{8}$ in. for stoping; (7) using only axial water-fed machines in stoping, and those mobile jackhammers. Increased research work into every branch of mining freed the manager and his staff from extraneous annoyances such as obtained before the strike. These points will be considered seriatim:

Discipline was at a low ebb prior to the 1922 strike. Miners lingered at shaft stations on the way to the working place, and then delayed perhaps a further half hour before starting the natives drilling. Other delays from lack of discipline were caused by pipe fitters, track layers, and trammers, none of whom exerted himself so as to insure that the machine driller was able to drill the maximum of holes per shift.

With discipline established, the trammer, who went underground two hours before the miners, started the machine natives drilling and saw to the safety of the working places. The miners, on arrival, went straight to their work, and the trammer was then able to concentrate on his tramming. The pipe fitter was reprimanded or dismissed if any undue delay arose from lack of application on his part. An estimated overall improvement of 20 per cent in the hours worked in the mines is no exaggeration.

The contract system in breaking ground is still the vogue in many mining centers throughout the world. Although in some instances local conditions may warrant its application, I fail to see why, in an up-to-date mine, any work need be given over to outside control, as is the invariable result of the adoption of this system. In South Africa the contract prices were invariably so high that the efficient miner made good money without effort and the poor miner made a very fair wage. The mine benefited little when easy-mining ground was encouraged. If the contract price were altered to suit, much ill feeling was caused, and complaints of unfair treatment were registered. Officials had no incentive to study and to improve on the method of drilling (the holes belonged to the miner-contractor and not to the mine); and thus much money was wasted because a large proportion of the miners conducted their mining operations unscientifically.

Payment by results, with the mine in complete control of every detail, was adopted in place of the contract system, and this was the most important factor in increasing underground efficiency on the Rand. The driller (native) was paid according to footage drilled. The gang-leader (white man) was paid according to the footage drilled by his men, receiving a bonus for clean mining, a bonus for safety, a bonus for footage drilled per square fathom, and likewise for explosive efficiency. Finally, the shift boss, given a bonus in similar manner for the results from his section, thus assumed complete charge of the miners. The good miner no longer received £80 to £100 per month, but was able to make £40 to £50 by hard work. The inefficient miner was lucky if his monthly check was £30.

Attention was now paid to making a miner out of the inefficient laborer. Stoping efficiency during the three years following the strike improved roughly 600 per cent. For example, in one mine the average fathoms stoped per machine per shift in 1921 was 0.5, and it was brought up to 3.5 fathoms; on an East Rand mine the same figures are 1.0 and 5.0. Lower explosive consumption accompanied the increased output.

The drilling of one hole per bench results in greater efficiency per foot drilled and per pound of explosive than if two or more holes are drilled in one locality and interdependent. If the reader has a doubt on the subject let him examine the efficiency figures of hand drilling of years gone by, when the two factors mentioned in the foregoing were definitely against the rigged-machine benches of many holes. The mobile jackhammer now used raises the efficiency of hand drilling, but is much cheaper, because one man can drill many holes at less cost per hole. On the Rand, jackhammers drill 4-ft. holes which are spaced 6 ft. apart along the stope face, so that no hole is dependent on another. The use of a hole director has tended to obviate the personal element in pointing holes. This simple appliance, causing no loss of time, enables the man who has difficulty in visualizing the bottom of a hole, before it is drilled, to point it perfectly. Thus the poor miner was able to earn better money than the £30 quoted, and the mines benefited considerably.

Theory definitely teaches that holes of small diameter produce better blasting efficiency than large-diameter holes. Practical investigation underground confirmed this, and indicated at the same time the reasonable limit in the reduction of the size of the bits of drill steel. Another point which practice confirmed is that much less rock is ground to powder for the same results with narrow-gage steel, which means a big saving in the consumption of air and steel. The practice developed into using $\frac{7}{8}$-in. hollow steel rotated by axial water-fed jackhammers. These latter not only render the sludging of such narrow-diameter holes feasible, but they increase the speed of drilling, and, of course, from a health point of view, they are desirable in any mine, siliceous or not.

The British and South African engineers, who now run the mines on the Rand, have always realized the value of research work, and what has been done in this respect since 1922 is highly creditable. It augurs well for the future of the Rand. And the day will come when the millions of tons of "Main Reef" averaging about 4 dwt. gold per ton, now lying dormant, will be brought into the ore reserves.

The Rand has certainly not yet reached its zenith.

Native dwellings, Roan Antelope, Rhodesia.

Earth Resistivity Surveying

G. F. Tagg

THE earth resistivity method is one of the simplest of the electrical type of geophysical methods of surveying. It is based on measurements of the apparent specific resistance of the earth. The presence of any body in the earth, the specific resistance of which differs from that of the surrounding material, will affect the apparent specific resistance to an extent dependent on a number of different factors. The method thus affords a means of detecting the presence of such bodies in the earth. It involves a system of measuring the specific resistance of homogeneous earth, described in a U. S. Bureau of Standards paper entitled "A Method

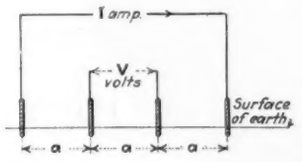


Fig. 1—A method of measuring the specific resistance of homogeneous earth

of Measuring Earth Resistivity," by F. Wenner. Four electrodes are driven in the earth in a straight line at equal intervals, a, as shown in Fig. 1, a current, I, is passed between the two outer electrodes, and the potential difference, V, between the two inner electrodes is measured.

Then if a is the electrode interval, and ρ is the specific resistance of the earth,

$$\rho = 2\pi a \frac{V}{I}$$

The fraction $\frac{V}{I}$ may be termed a resistance and designated R, and then

$$\rho = 2\pi a R.$$

Inasmuch as only under special conditions can the earth involved in the measurements be considered homogeneous, the value of ρ obtained by using these formulas is only an apparent specific resistance, and will be affected by the extent of the non-homogeneity and the electrode separation.

Apparatus has been devised for measuring both V and I, and also for measuring the resistance R directly. This will be described later.

Results may be interpreted by three different methods. These are the empirical method, the resistivity map method, and the method of theoretical analysis.

Empirical Method—The method used up to the present for interpreting the results is an empirical one and is not confirmed by any theory. Suppose a series of tests are made at one point, keeping the center of the electrode system at that point but varying the electrode interval. Then a series of values for the apparent specific resistance will be obtained. If a curve of apparent specific resistance against electrode interval be plotted, a sudden change in curvature is taken as indicating the presence of a body which has a conductivity different from that of the surface medium, and the method assumes that this body lies at a depth below the surface, equal to the electrode interval at which the change in curvature occurs. This interpretation is based on tests carried out by Gish and Rooney over a filled-in ravine and also on a lake. In the curves representing their observations, they obtained certain changes in curvature at points equal to the depth of the ravine or lake, and concluded from this that the body of earth included in tests of this type is limited substantially in all directions from the line joining the potential electrodes to a distance equal to the electrode interval.

Most of the users of the method have adopted this way of interpreting their results, but as a majority of the tests appear to have been carried out in regions where the substructure of the earth was known, they cannot be called impartial. From theoretical considerations, sudden changes in curvature in the curves representing results obtained where horizontal strata occur over one another would appear to be unlikely, and if they are found their explanation must be sought in some other type of geological formation.

Resistivity Map Method—Suppose a series of measurements be made over an area with the electrodes at a fixed distance apart, the center of the electrodes being moved from point to point. Then it can be shown that in the majority of tests, provided the electrode separation is somewhat greater than the depth of the body it is desired to measure, the conductivity—that is, the reciprocal of the specific resistance—is proportional to the depth of the body. Thus the practice is to plot lines of equal conductivity on a map, and to interpret these as contour lines of depth. If, then, the depth be absolutely determined at one point, the depth at any other point may be obtained by proportion. This method of interpretation has yielded satisfactory results.

Theoretical Method—For certain simple formations of earth structure, calculation of the value of the apparent specific resistance for a given set of values of the specific resistances of the earth, electrode separation, and depth is possible. This has been done in using the method for a single horizontal stratum, and a method of interpretation based on this theory has been devised. This was used in an experimental survey at Cleeve Common, Gloucestershire, England, and proved satisfactory. It is being developed to deal with more complicated forms of earth structure.

Two forms of apparatus can be used in carrying out measurements by this method, the first being the potentiometer-milliammeter equipment, originally designed and used by Gish and Rooney, and the second, the "Megger" ground-resistance tester, developed by Evershed & Vignoles, Ltd., London.

The potentiometer-milliammeter apparatus originally used was employed by Gish and Rooney for measuring the specific resistance of earth in connection with investigations made for the Department of Terrestial Magnetism at Washington. It consisted of a poten-

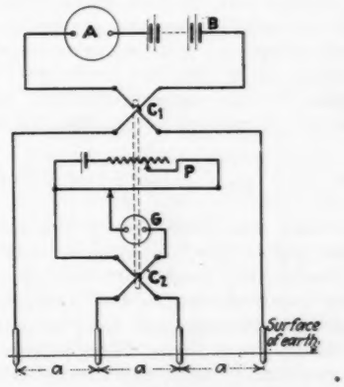


Fig. 2—Connections used in the potentiometer-milliammeter equipment

tiometer, milliammeter, a battery, and a set of reversing commutators. A diagram of connections is given in Fig. 2, in which A is the milliammeter, B the battery, P the potentiometer, G the galvanometer used with the potentiometer, and C_1 and C_2 are the reversing commutators mounted on a common spindle so that the reversals are synchronized.

Immediate recognition was made of the fact that the employment of direct current would introduce a number of errors such as those traceable to electrolysis, back emf., and stray cur-

rents, and the use of alternating current in the soil was necessary. On the other hand, to use direct-current instruments for the actual measurements was more convenient, particularly as direct current could be obtained from a portable source such as a battery. These considerations led to the inclusion of the reversing commutator, C_1, shown in Fig. 2, to convert the direct current from the battery after it has passed through the milliammeter to alternating current, and to the inclusion of C_2 to rectify the current produced by the potential difference between the two inner spikes, so that balance could be obtained on the potentiometer. Use of these commutators introduces a correction factor into the result if the correct value of R is to be obtained, since the current is definitely switched off for a short period in every half-cycle of the alternating current. The milliammeter reads the current I passing between the outer electrodes, while the potentiometer gives the potential difference V between the two inner spikes. The ratio $\frac{V}{I}$ after modification by the correction factor caused by the commutators gives the resistance R. Apparent specific resistance may then be calculated.

The four spikes or electrodes will each have a resistance to earth, and if the ground is dry or sandy these resistances may easily be of the order of several thousands of ohms each. The resistances of the two outer electrodes are included in the main current circuit, and reduce the testing current and the voltage drop between the two inner spikes in the same proportion. They do not therefore affect the value of the ratio $\frac{V}{I}$. If these resistances are so high that they reduce the current flow appreciably, this may be overcome by increasing the battery voltage. The two inner electrodes are connected to the potentiometer, and since no current is taken from them when balance has been obtained, their resistances to earth do not affect the value V measured on the potentiometer. If the resistances of these electrodes are large, they may reduce the sensitivity of the potentiometer, but this will usually not be serious. Obviously, the main drawback to this apparatus is its bulk, as several instruments and a battery are required, in addition to the electrodes, connecting cables, and the necessary complementary apparatus.

The "Megger" ground resistance tester is a combined ohmmeter and generator of specialized type, so designed that alternating current is supplied to the soil section of the testing circuit to overcome difficulties caused by electrolytic back emf., and direct current to the measuring part of the testing circuit to allow the use of the direct-reading moving-coils ohmmeter with its large working force and uniformly divided scale.

In construction, the ohmmeter embodies two coils mounted at a fixed angle to one another on a common axle and swinging in the field of a permanent magnet. The axle carries a pointer moving over a scale marked in ohms. A current proportional to the total current flowing in the testing circuit passes through the current coil, while the potential coil carries a current proportional to the potential drop across the resistance under test. The coils are so wound that the resulting forces oppose one another.

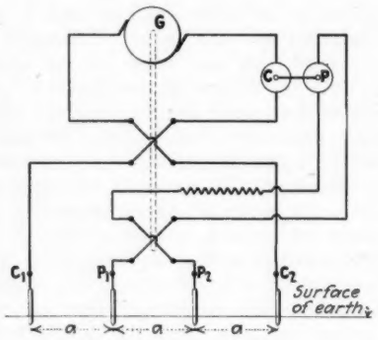


Fig. 3—Diagram of connections when using the Megger

The final position of the moving coils —and hence that of the pointer—depends on the ratio of the potential drop to the total current, and the instrument is therefore a true ohmmeter, giving readings in ohms, which are independent of the applied voltage.

Connections are as shown in Fig. 3. Direct current from the generator G passes through the current coil C of the ohmmeter to a rotating current reverser, and to the terminals C_1 and C_2 of the instrument, which are connected to the two outer electrodes. The potential coil P of the ohmmeter obtains its supply from the terminals P_1 and P_2, which are connected to the two inner electrodes. As this supply is taken from the soil section of the current circuit, and is therefore alternating, it must be made unidirectional before passing through the potential coil. A commutator mounted on the same shaft as the main current-reverser, and synchronized with it, is therefore interposed as a rectifier between the terminals P_1 and P_2 and the potential coil. In this manner the current and potential coils of the ohmmeter are both supplied with direct current, and the soil section of the testing circuit is supplied with alternating current.

The instrument is so calibrated that the value of the resistance R is read directly on the scale, and thus the apparent specific resistance can be at once determined.

Resistances of the outer or current electrodes will not affect the result, for the reason already explained in respect to the potentiometer-milliammeter equipment. The resistances of the potential electrodes will, however, have an effect, as current is taken from the potential electrodes and any large resistances at the potential electrodes will cause the instrument readings to be too low. In calibrating the instrument, a certain value is allowed for the total electrode resistance, and a simple correction can be applied to the resistance obtained to give the correct value. Thus if

R_a = resistance read on scale in specific resistance test;

p = total electrode resistance for which allowance is made in calibration;

l = total internal resistance of potential circuit of instrument;

$p_1 + p_2$ = actual potential electrode resistances;

R = correct value of resistance.

The true value of the resistance being measured, R is given by the formula

$$R = R_a \left[\frac{l + p_1 + p_2}{l + p} \right]$$

Values of l and p are given on the instruction card supplied with the instrument.

Resistances of the potential electrodes may be measured by the instrument itself by connecting the terminals P_1 and C_1 together, and to one potential electrode, the second potential electrode being connected to the P_2 terminal, and one of the current electrodes to the C_2 terminal. The instrument then reads the resistance of the electrode directly on the scale.

The Megger at present used for this class of work has four ranges of 0-3, 0-30, 0-300, and 0-3,000 ohms.

In all surveys carried out up to the present using both the potentiometer-milliammeter equipment and the Megger, discrepancies are alleged to have been recorded between the values obtained by the two sets of apparatus. Fundamentally, there is no reason why the results obtained should not agree.

In the survey previously mentioned which was carried out at Cleeve Common, no difference was discovered between the results obtained with the two sets of apparatus.

Interpretation of the results is the most difficult part of the whole procedure. If the mathematical theories can be developed to cover more general tests of ground structure, they will provide a very accurate means of interpreting the results, and a means which does not depend on the judgment of the observer.

Tapping Another Source of Manganese

A scene on the Manganese Corporation's properties

ONLY the future will show whether or not the plans of the Manganese Corporation in South Africa will be delayed by the conditions now prevailing in the manganese industry throughout the world as a result of the Soviet government's efforts to push the sale of Russian manganese ore. This corporation was organized with a capital of £1,000,000 in 1929 by the British Swiss International Corporation, Ltd., to exploit the manganese deposits discovered in the Postmasburg district of South Africa in 1922.

At first, the Postmasburg deposits were not taken seriously, and only after thorough examination, by the interests named, was exploitation begun. Ultimately, the Corporation expects to furnish 300,000 to 400,000 tons annually.

According to Paul M. Tyler (*E.&M.J.*, Feb. 8, 1930, p. 129) the company has paid £302,000 to the government of South Africa for building a railroad which has been finished for some time. Shipments from this field have already been received in the United States.

A large tonnage of hard ore, averaging 49 per cent manganese and running low in sulphur, silica, and phosphorus, is said to have been proved. Some running as high as 55 per cent manganese is also to be had.

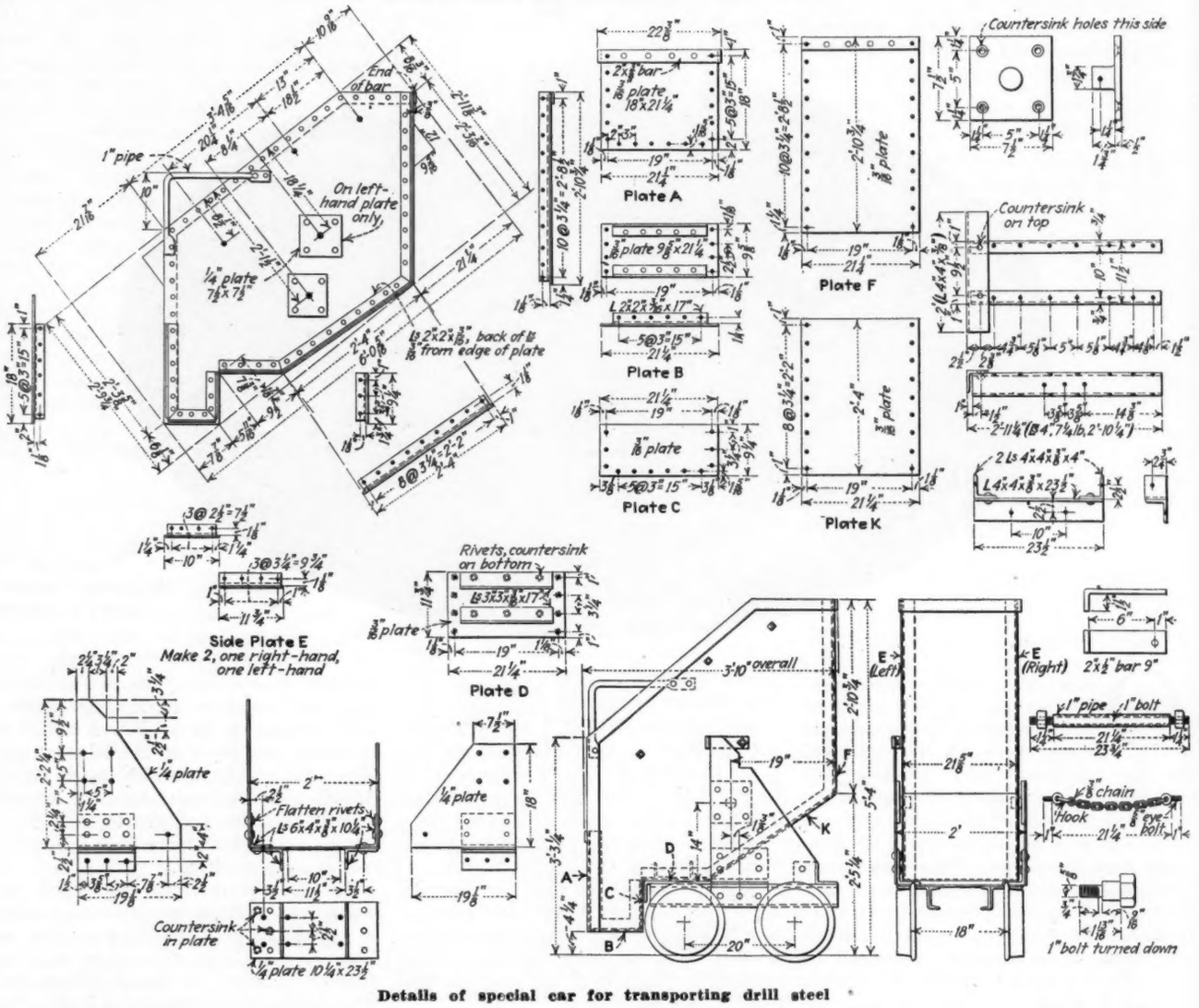
Mining costs will be low, according to Mr. Tyler, but railroad freight, estimated at 16s., and ocean freight, estimated at 20s. 6d., and other charges incident to delivering the ore to European or American consuming centers, may amount to 18 or 20c. a unit. It is said that power is available at the mines for making possibly as much as 300,000 tons of ferromanganese, should this be desirable and provided the saving in freight will more than cover the cost of local smelting.

The Corporation has developed the deposit and started operation of a plant to crush about 1,000 tons a day to 5-in. size. Later, this plant will be doubled in size. A model village for white employees is also being built. The loading plant will be at Durban. Three photographic reproductions are shown on this page by courtesy of the *Empire Mining Review*, Johannesburg.

An open cut in manganese ore, Postmasburg district

The Postmasburg ore is hard and of good quality

Special Car for Transporting Drill Steel In Underground Workings



Details of special car for transporting drill steel

TO OVERCOME difficulties encountered with the standard transfer car of the rigid-body type, when transporting long drill steel through low and small drifts, a special car, adjustable to prevailing underground conditions, has been developed by the United Verde Extension Mining Company, at Jerome, Ariz. The important feature of this car is the suspension of the car body on two pivots. This arrangement permits the car to be tipped to a horizontal position, thus making possible the handling of drills of considerable length in small drifts or tunnels.

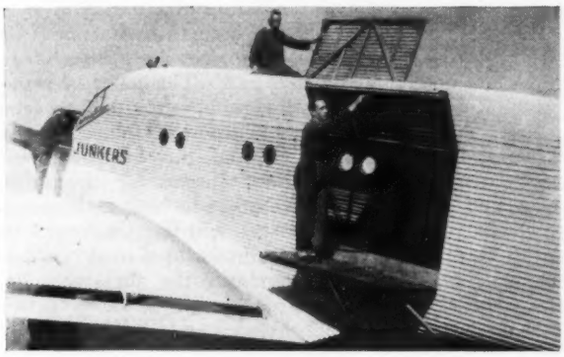
As is shown in the accompanying drawing, the car consists of a four-wheeled truck, with a bolted-on frame supporting the body, and the car body. Six-inch channels, 6x4x$\frac{3}{8}$-in. angles, and $\frac{1}{4}$-in. plate are employed for the construction of the truck frame and body support, and $\frac{3}{16}$-in. plate, with $\frac{3}{8}$x2-in. bars and 2-in. angles for reinforcement, are used for building the car body. The two mild-steel pivots are drop-forgings, and are finished in the lathe to proper size. A 1-in. pipe spacer is provided to insure additional stability of the car body. Steel is kept from shifting by two small chains bolted onto the sides of the car body. A simple locking device made of $\frac{1}{2}$x2-in. bar holds the car body rigid in either an upright or a horizontal position. Hand tramming is facilitated by two handle bars made of 1-in. pipe that are riveted to the sides of the car body.

Direct-current motor windings, coated with a bright orange-colored enamel instead of the usual black finishing coat, are now being produced by the Reliance Electric & Engineering Company, Cleveland, Ohio, to obtain steadier operation and longer life. The colored enamel serves to aid in detecting dirt and affords, in addition, an excellent insulation.

General Electric has developed a new line of electric brazing equipment of general industrial application. Brazing is effected with an alloy by means of heat generated by flow of electricity through carbon blocks. The equipment consists of a transformer, a foot switch, and tongs. A typical unit involves a 5-kva. transformer weighing 45 lb. and having a 220-volt primary and an eight-volt secondary. Advantages claimed are that less time is required, brazing is less expensive, and the conductivity, mechanical strength, and durability are higher than those of the lead joints.

INDUSTRIAL PROGRESS

New Airplane Will Carry Heavy Freight Long Distances



Large openings in side and top of freight room facilitate loading

AFTER much experience with the Bremen-type Junkers W-33 airplane, Junkers company has brought out a new model, Ju 52, for hauling heavy freight long distances. Compared with Model W-33, the new plane has a capacity of 3,000 kg. (6,612 lb.) instead of 1,200 kg.; a loading room of 22 cu.m. (777 cu.ft.) volume instead of 4.5 cu.m., and a deck space in this room of 10.5 sq.m. (113 sq.ft.) instead of 3.75 sq.m. It has a 700-hp. motor, more than double the horsepower of the older model. A fuel supply of 1,000 kg. (2,204 lb.) may be carried, giving a cruising range of 1,500 km. (932 miles) for 9½ hours. By a slight change, another 1,000 kg. of fuel may be carried without reducing the space available. The ship weighs 4,000 kg. and will carry a load of 3,000 kg., including fuel. Maximum speed and traveling speed are 194 and 160 km. per hour, respectively, and fuel consumption is 115 kg. per hour.

The assumption that only materials having a high value in proportion to their volume can be carried by airplane is true only for European conditions, and not always then, says the builder. Practical experience shows that heavy materials and large single pieces may be handled to advantage. One of the illustrations presented herewith shows the large freight room of the Ju 52. Another shows the conveniently large openings provided in the sides and top

Freight room of new Junkers Ju 52 airplane

to permit quick loading. The main openings are 1,700x1,300 mm., (5.6x4.3 ft.) and 1,700x900 mm. respectively. The main freight room, 1.9 m. high, may readily be converted into a roomy cabin, shown in the cut, with the necessary auxiliary rooms.

■ ■ ■

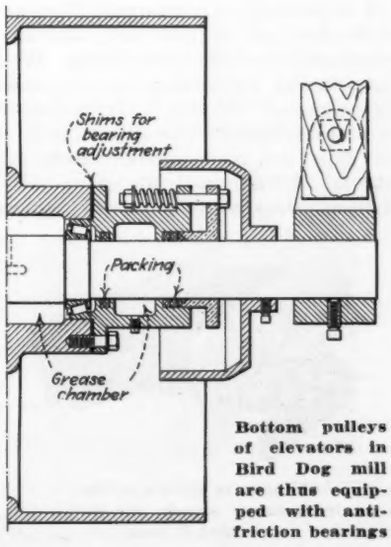
Varied Developments in Equipment Design

Scope Includes Slushing, Excavating, Haulage and Pumping; Also Crushing, Screening and Concentration

AN ELECTRIC double-drum hoist for slushing, or for any dragline scraper work within its capacity, has been introduced by Ingersoll-Rand, New York. It is known as Size 215, and its rated capacity, vertical lift, is 2,000 lb. at an average rope speed of 240 ft. per minute. It has a 15-hp. motor, and its dimensions permit it to pass through a 25x25-in. opening with the rope guide removed. The drum will take 300 ft. of ½-in. rope or 500 ft. of ⅜-in. rope. It may be had with either a.c. or d.c. motor.

Boot Pulley Timkenized

All elevators in the new Bird Dog mill of the Commerce Mining & Royalty Company, Miami, Okla., are equipped with Timken-bearing bottom pulleys. One of these pulleys is shown herewith. In this design the shaft does not rotate and the pulley bearings are lubricated by an oil line through the shaft, which can be run to the floor, making lubrication easy. The pulley hubs are well sealed to prevent water and grit from entering the bearing chamber.

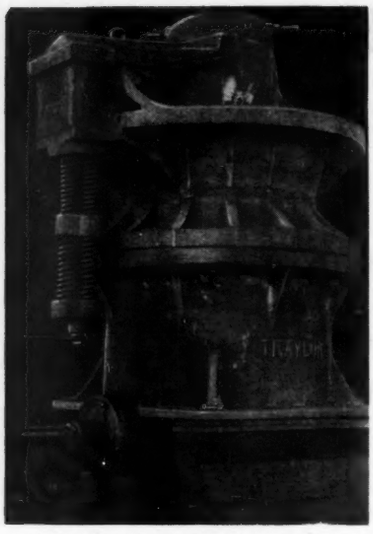


Bottom pulleys of elevators in Bird Dog mill are thus equipped with anti-friction bearings

Two New Excavators of Small Size Have Been Developed

Among excavators recently developed is a new ⅜-cu.yd. machine, Model 200-A, introduced by Harnischfeger Corporation, Milwaukee. It is built with a generous use of alloy steels and is light enough to meet the demand for a convertible excavator weighing under 14 tons. It is powered with a four- or six-cylinder 45-hp. gasoline engine and is mounted on crawlers in accord with present-day demand.

Another excavator, known as the Wright, is a full-revolving, fully convertible ½-yd. shovel announced by the Pontiac Tractor Company, Pontiac, Mich.



Principal feature of this new crusher can be applied to any existing gyratory

A New Reduction Crusher

A machine that the manufacturers think will revolutionize fine crushing is the new reduction crusher developed by the Traylor Engineering & Manufacturing Company, Allentown, Pa. A preliminary reference to this machine was made in the Annual Review Number (Feb. 9 issue), page 136. It is styled the "Type TZ," is designed with extra-heavy proportions and built in six sizes capable of an hourly production ranging from 15 tons, ½-in. ring size, to nearly 1,000 tons 2½-in. ring size. The crusher is of the gyratory type, with a bell-shaped crushing head and concaves that are sharply convexed from top to bottom of the bowl. This design is said to eliminate choking, because the normal choking point of a standard-type gyratory crusher (at the bottom of the head) has been moved upward to a point where the tendency to choke disappears. This results in a higher rate of output and a finer, more uniform product with a reduced amount of undersize. The crusher is designed with a balance lever and tension springs to support head and shaft, the springs being adjusted to a tension sufficient to withstand the downward thrust of the head due to crushing, and yet to allow the head to be depressed to a liberal degree. The head may be adjusted accurately, during operation, for varying the size of product, the adjustment range in any one machine being 50 per cent. That is, if the original coarsest setting is 1½ in., the machine may be adjusted to break to ¾ in. or anything between these limits. No feeder is required or should be used, as the crusher operates most efficiently when flooded. The new crusher is said to possess all of the valuable features of the Traylor Bulldog gyratory, including the bar type non-weaving spider, the stocky non-deflecting main shaft, the self-aligning eccentric bearing, the automatic lubrication system, with all elements sealed, and dust-sealing.

An advantage of the new design, according to the manufacturer, is that at reasonable cost it may be applied to any existing gyratory crusher of any make and may be installed with or without the balance lever and tension springs. The effect of such conversion is to increase enormously the capacity at a smaller product size. The increase averages 50 to 75 per cent. and it is said there are instances where the former capacity has been doubled with the product size less than that of the previous minimum setting.

Vibrating Screen Improved

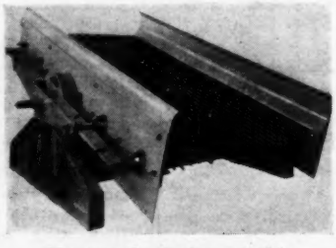
An accompanying cut shows an improved Gyrex screen made by the Robins Conveying Belt Company, New York. It shows that the horizontal leaf springs run lengthwise with the screen instead of resting in the usual shackle, and are now rigidly fastened to a transverse leaf spring set on edge and bolted to the live frame. One purpose of these transverse springs is to center the frame on the shaft and thus remove from the eccentric bearings any side-thrust load. This arrangement is said also to do away with the need for shackles with their attendant maintenance expense. Another change has been made in the vibrating unit. The counterweight is in the form of an overhanging arm and is in the same plane as the eccentric bearing. All bending stresses due to centrifugal force are eliminated. This counterweight also permits of close compensation for the unbalanced load of the live frame. There are no escaping vibrations.

An Explosion-Proof Push Button

A new oil-immersed explosion-proof push button has been introduced by the Electric Controller & Manufacturing Company, Cleveland, Ohio. It is made of die-cast aluminum, an alloy of high tensile strength being used. It is said to be able to withstand pressures set up by explosions of gasoline and similar vapors. Use of the aluminum alloy cuts down the weight materially as compared with cast iron.

A Chain Hoist of Aluminum

A chain hoist of aluminum alloy, said to be the first in this field, has been introduced by Chisholm-Moore Hoist Corporation, Tonawanda, N. Y., under the name of "Al-Lite." It is said to be a third lighter than other hoists of this type, and to have fewer parts and unusual strength and efficiency. Only certified Alcoa alloys are used. Among its features are hardened and ground ball-bearings, planetary-type gears, and a dust-proof housing. It will stand a 50 per cent overload. One man can carry it and hang it in place.



Changes in design of this vibrating screen are described in text

Some Motor Haulage Accessories

Westinghouse engineers have improved the design of the cable guide for gathering reels on 250- and 500-volt Baldwin-Westinghouse mine locomotives. The design differs from previous ones in that it is of a three-piece construction consisting of a hardened-metal center and two outer Micarta insulating rings, whereas former designs were constructed as a unit. The inner or center portion of the guide is the only wearing part. The construction permits the operator to replace the hardened-metal center without discarding the entire guide, thereby reducing maintenance costs. Also the wearing portion of the guide maintains a polished surface, thereby eliminating the abrasive action and increasing the life of the insulation.

The increasing use of 6-0 trolley wire for handling heavier locomotives and larger loads has caused the Ohio Brass Company, Mansfield, Ohio, to re-design its Bulldog trolley clamp, to give it larger and heavier parts. Three-inch jaws, having a greater opening, have been provided to accommodate the larger wire. These are heavier and the gripping nut and hinge are more substantial, the result being that the clamp has greater gripping power and strength. This grip may be adjusted to any extent deemed advisable, to allow the clamp to release the wire under excessive stress. As in former models, the jaws open and close by positive action brought about through the vertical motion of the head nut. The entire clamp is made of so-called Flecto malleable iron, hot-dip galvanized. This company, in the March issue of "O-B Haulageways," advocates periodic testing of return circuits, to disclose bad joints. The power losses in the return circuit, often due to inferior bonding, says the company, can easily amount to three or more thousand dollars per year. One cannot assume that a bond once installed will last forever without further attention. An instrument which seems to be most satisfactory for bond testing is a duplex millivoltmeter, or two millivoltmeters mounted on the same panel, which answers the same purpose. The article describes the method of testing.

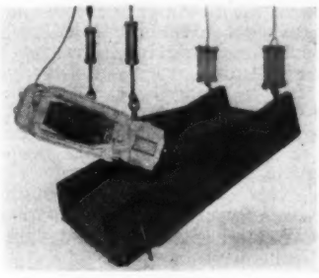
For Better Rail Bonding

A device to secure more efficient electric haulage by improving the rail bonding is the new type Tiger-Weld power bond developed by the American Steel & Wire Company, 208 South La Salle St., Chicago. This bond has solid steel terminals flash butt-welded to the copper conductor. Thus, the welder is relieved of the difficult task of welding the wires to the terminal, this assuring easier, quicker welding and a good job. By this newly developed manufacturing

process, the full cross-section of the conductor is maintained throughout its entire length and the possibility of wires becoming "crushed" or "necked down" is eliminated. Likewise, there are no abrupt deformations to induce concentration of vibratory stresses, so that a high resistance to fatigue always prevails, insuring longer mechanical life.

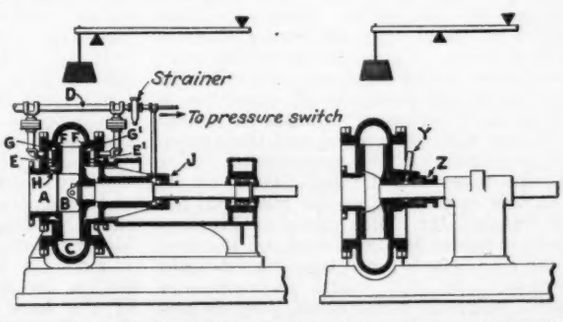
Something New in Pumps

To overcome the high maintenance, low average efficiency and the discharge-pressure limitations inherent in ordinary dredge pumps handling ash-laden water, the Allen-Sherman-Hoff Company has developed a so-called "Hydroseal" pump, which is built along unconventional lines. Several hundred of these pumps are said to be in service. In view of their success in pumping ashes, the manufacturers feel that the pump should have wide application in pumping sand, gravel, ore, rock, or in handling any



This vibrating conveyor screen may be run flat

abrasive-laden liquid. In pumping, the operation of the unit is identical to that of any standard centrifugal pump. This is shown in the left-hand diagram in the cut below. Abrasive-laden liquid enters the suction at *A*, passes to the impeller *B*, thence to the volute *C*, to the pump discharge. As compared with the conventional pump design, the new unit has the following interesting features: The design is such that the annular clearances *F* and *F'* are continuously filled with clear water flowing into the volute *C* and not water laden with abrasive material flowing in the opposite direction. Thus, the abrasives being pumped cannot wear the outside of the impeller shrouds or the side plates *G* and *G'*. Leakage through the sealing ring *H* is of low-pressure clear water only and not high-pressure abrasive-laden water, this permitting the pump to retain its initial efficiency and discharge head indefinitely. The clear water introduced at *E* and *E'* need not be of a pressure higher than the pump discharge pressure, as is necessary with gland water introduced at *Y* in conventional pumps. The gland *J* merely holds against the low-pressure clear water to the annular bearing, so that little attention need be given it. The gland *Z* in conventional pumps must hold against water of higher pressure than that of the pump discharge. Abrasive water cannot leak out along the shaft of the new pump. There is almost no overhang of the impeller shaft beyond the bearing. The diagram above each pump and cross-section presents a clear analysis of the forces tending to produce shaft whip in the new pumps and that of conventional design. Since the bearings in the new pump are located so that there is almost no shaft overhang, the peripheral speed of the impeller can be increased far above anything hitherto possible. Likewise, it is practical to produce pressures as high as 150 lb. per square inch and still maintain high efficiencies. The amount of clear sealing water is minimized and controlled, an advantage where such water is expensive or where it is important to maintain maximum concentration. The clear sealing water in the new pump controls the electricity or steam supply for its drive so that it is impossible to start or operate the pump unless clear water is available. Thus, careless operation or failure of the water supply cannot ruin the pump. The new pump may be had in both horizontal and vertical units.

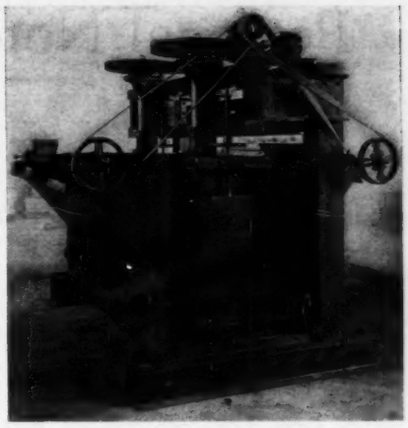


Left, diagram illustrating design of new type centrifugal pump. Right, the conventional design. See text.

New Practices in Screening

The so-called "Trayco Conveyan-screen," developed by the Traylor Vibrator Company, Denver, Colo., is said to introduce some new practices to the art of screening. The power or vibrator unit is mounted above the screen sash and imparts its vibration at an angle to the screening plane, thus producing a sharp screening action as well as a conveying motion. Thus, the screen can be operated practically flat, doing away with the necessity of setting the screen at an angle so that gravity will cause the flow of the material across the surface.

The claim is made that this screen produces more accurate sizing because the angle of vibration causes the material to keep in closer contact with the screen cloth. The vibrator is suspended so that all vibration is absorbed within the unit. The screen is furnished in four standard widths and in any desired length. On single- and double-deck screens up to 6 ft. long one vibrator is used. Between 6 and 12 ft. of length two vibrators are used, and beyond this length three or more.



A Flotation Machine Designed for Low Head Room

Tests have been completed on a new Denver "Sub-A" (Fahrenwald) flotation machine of low-head design, which permits installation in close quarters. One-third less head room is required, according to the Denver Equipment Company, Denver, Colo., the manufacturer. This machine has all-steel bents which are practically vibrationless. The tank can be of welded-steel design, or if this would affect the metallurgy a wood tank can be used with the steel bents. Greater shaft rigidity is assured through decreased length. The lower bearing has been changed to an SKF self-aligning ball bearing, which, together with the upper radial-thrust bearing, eliminates any possibility of bearing trouble. The bearings are sealed against dust. The unit ball-bearing shaft-assembly naturally requires less horsepower. Construction of the machine is such that an unobstructed view may be had of the entire length, which, together with the increased light, reduces the attention required to a minimum. The cut shows the end view of a two-cell machine used for experimental purposes. Ten-cell machines have been in actual service for some months.

INDUSTRIAL NOTES

The Foster Wheeler Corporation has entered the field of boiler manufacture by acquiring the plant and business of the D. Connelly Boiler Company, Cleveland, Ohio.

Colorado Fuel & Iron Company has erected at its Pueblo Works a new building which will house, among other equipment, that necessary for making forged grinding balls of several different diameters.

BULLETINS

Conditioning (for flotation) and Agitating — Denver Equipment Company, Denver, Colo., Bulletin No. 2903.

Portable Mills—Denver Equipment Company, Denver, Colo., Bulletin 2907, entitled "Insuring Mining Investments With Denver Portable Mills."

Personal Notes . . . Comment . . . Criticism

Among Our Contributors:

ALAN PROBERT, who describes an improved and proved method for sieving ore pulp, has been associated with concentrator work at properties of United States Smelting, Refining & Mining. Since graduation at the university of California he has had a broad experience, ranging from plant operator to mill superintendent. In 1928 he was appointed a member of an advisory committee to promote joint co-operation in mining and metallurgical research among the U. S. Bureau of Mines, the University of Utah, and the mining industry.

BERNARD BERINGER, now with Cerro de Pasco, at Morococha, Peru, is the author of "Underground Practice in Mining," published in 1928. He is a graduate of the Camborne School of Mines, with which was long associated as principal his father, J. J. Beringer, author of a standard textbook on assaying. Bernard Beringer, since the War, has followed his profession in India, Nigeria, and Spain, in addition to four years' experience in the district of which he now writes—the Witwatersrand of South Africa.

C. N. SCHUETTE, a consulting engineer of San Francisco, Calif., is a graduate of the University of California and an authority on an art that, until recently, has been somewhat neglected—the metallurgy of quicksilver ores. He is co-author with L. H. Duschak of "The Metallurgy of Quicksilver," published in 1925 by the U. S. Bureau of Mines, and of several other publications of the Bureau.

MORRIS J. ELSING, a consulting engineer with headquarters at Tucson, Ariz., who is contributing a series of articles on mining methods, was formerly geologist to Copper Queen Consolidated, and later in charge of the geological department and of the mining operations of Cananea Consolidated, prior to acting in a consulting capacity to that corporation.

G. F. TAGG, of Hounslow, England, is an electrical engineer specializing in geophysical prospecting by electrical methods.

F. G. HAWLEY is chief chemist for International Smelting, at Inspiration, Arizona.

■ ■ ■

Torna Barrera, professor of geology at the National University, and chief of the division on non-metallics in the Institute of Geology, Mexico City, has been awarded a Guggenheim fellowship. He will work at Howard University, with special reference to non-metallics and geophysical exploration.

H. K. Faye, formerly freight traffic manager for the Western Pacific Railroad, has been appointed freight traffic manager for Phelps Dodge and subsidiaries.

Marshall W. Tuthill, of Tuthill & Company, has become associated with W. K. Johnson & Company, members of the New York Stock Exchange. He will specialize in mineral and metal securities and metal exchange accounts. The affiliation involves no change in the affairs of Tuthill & Company, which will continue as general agent in the United States for Mansfeld Mining & Smelting, and is now located at 40 Wall St., in New York.

Major General Lyle Brown, Chief of Engineers of the United States Army, will be the commencement speaker at the Michigan College of Mining and Technology at Houghton on May 21.

Robert C. Stanley, president of International Nickel, at the recent annual meeting of that company, pointed out to stockholders that with adequate ore reserves the company has become a manufacturing rather than a mining enterprise. He said that in a period of depression the management had thought it wise to increase its expenditures on research in the development of new uses for nickel.

C. S. Anderson, whose article on the Vermont copper district appeared in the March 9 issue, calls attention to the fact that his address is South Strafford, Vt. He adds that the Marcy ball mill, referred to on page 209, is a No. 86 mill, the Cameron sinking pump a No. 7 unit, and the ratio of solids in the ball-mill discharge, given on page 210, 70 to 75 per cent.

R. C. Allen, of Cleveland, Ohio, vice-president and manager of mines for the Oglebay, Norton Company, which has a contract to act in an advisory capacity in the development of iron-ore fields in Russia, recently stated that "the new economic policy in Russia is virtually that of state capitalism. Methods employed do not differ materially from those practiced by great corporations in this country. Development of the iron-ore resources of Russia is controlled by one great corporation, all planning is done by government officials in charge of these various groups, and the capital is furnished by the government. The Soviet takes the United States as its example in building up the industrial life of Russia," Mr. Allen stated. "Tremendous progress has been made on the five-year plan, much more than was thought possible," he said, "factories have been built, railroads constructed, schools and colleges established, and welfare agencies started."

Dr. Rudolph Wuensch, of the engineering staff of Fried. Krupp A.G., returned to Germany on April 1, after a visit to United States and Canada.

Alan Bateman, professor of economic geology, Yale University, and editor of *Economic Geology,* lectured recently at Columbia University on "Some Problems of the Mineral Deposits of Southern Africa."

S. E. Stein, formerly manager of the mining department of the Southwestern Engineering Corporation, of Los Angeles, has resigned to undertake private practice in New York. H. B. Menardi succeeds Mr. Stein with Southwestern.

Endicott R. Lovell, assistant superintendent of the Calumet & Hecla smelter, has returned to Hubbell, Mich., after a trip including England, France, Germany, and Italy.

James F. McClelland, of New York, formerly a vice-president of the Chemical Bank & Trust Company, has been appointed a vice-president of Phelps Dodge. Mr. McClelland, a mining engineer, was formerly a professor of mining at Leland Stanford, and later at Yale University.

George S. Goodale, formerly superintendent of Mayflower-Old Colony, in the Michigan copper district, who has been in the employ of the Soviet government for the last nine months, has been promoted to the post of chief engineer of the projecting branch for all nonferrous mining undertaken by the Russian government.

Fred C. Carstarphen, of Denver, Colo., who will be recalled as the author of "The Bee-Line Principle of Rope-Tramway Location" and "Truck or Cableway? A Comparison of Costs," which articles appeared in *Engineering and Mining Journal* last year, recently addressed the Colorado Society of Engineers on "Silver and Its Economic Problem." In the course of his address Mr. Carstarphen said: "It is plain that the analysis that we have pursued in primary economics does not sustain any sound argument for trying to reduce our silver stocks under the guise of a loan of 200,000,000 oz. to China to help rehabilitate her foreign trade. China rightfully gives this proposal the go-by. It will be some time before other uses will be found for silver other than the electrical and film divisions of industry. The hand-writing has been on the wall for silver ever since 1816 in England, 1834 in the United States, 1873 in Germany and the Latin Union, and 1893 in India. It is the duty of the friends of silver to suppress their sentiment, their feelings concerning the passing of this noble metal, and set to work to develop a place for it in the arts and commerce of the world. I hope to see a National Silver Institute organized for this purpose. Silver will always have a place in subsidiary coinage, it will always be a byproduct of the mining of the base metals, and its industrial demand must be stimulated as a proper step to aid its price recovery. . . ."

Placer Gold in Peru

To the Editor of "E.&M.J.":

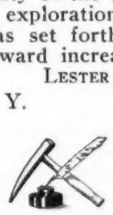
In perusing Mr. H. D. Keiser's description of "Placer Gold in Peru," in your issue of March 23, 1931, relative to the concessions granted by the Peruvian government to Mr. Clarence Woods which cover the Inambari River region, I note that the gold possibilities of the supposed Inca sources are again brought to the attention of the mining world.

It may be of interest to mention, as a note of record regarding "opportunity for the application of modern mining methods," that the gold alluvials of the region have been sporadically exploited, within the past forty years, by hydraulicking and dredging operations. One effort was that of the Sandia Mining Company, formed in Ohio, in which the water of the gold-bearing stream was diverted through a cut-off tunnel. A later (about 1911) hydraulic operation was that of the Aporoma Goldfields, Ltd., of London, which has been inactive for many years. The Kachi Kachi deposit was less actively exploited. The three operations are in Mr. Woods' concession. About twenty-five years ago the Inambari Gold Dredging Company, Ltd., an English company (to which English and American capital was subscribed) built a dredge on the river that was washed away by a torrential flood before completion. (Floods prevail each rainy season.) A second dredge was floated lower down the river in 1911. It operated for several months and was then abandoned because of the non-commercial results obtained. This second attempt involved an outlay over £80,000—a total loss when the dredge sank. The "exploitation of the deposits by modern methods" has therefore not "been delayed by the inaccessibility of the region."

The general exploration and exploitation scheme, as set forth, is an ambitious effort toward increased gold production.

LESTER W. STRAUSS.

New York, N. Y.



Geophysics in the Mining Schools

To the Editor of "E.&M.J.":

The time when geophysics occupied a somewhat doubtful position in the mining and oil industries has long since passed. Today, no experienced engineer will question the usefulness of training in geophysics and the application of the science to problems arising in exploration work. Recently a trend to establishing courses in geophysics and applied geophysics at various universities and schools has been noted. The California Institute of Technology has invited Dr. Gutenberg to be professor of geophysics and seismology. The University of Pittsburgh, Massachusetts Institute of Technology, and Columbia University, as well as various Mid-Western schools, are giving courses in geophysics or expect to give them in the near future. Likewise the University of Mexico, in Mexico City, is giving courses in seismology, tortion-balance work, magnetic prospecting, and other branches of the science. Only recently I met a few young (Mexican) geologists who were leaving the United States to study geophysics in the capital of their country.

Unfortunately, there is a marked tendency to neglect the teaching of applied geophysics in favor of the pure science. A harmful tendency is also noticeable among the mining schools, with their somewhat weak and overloaded departments of mathematics and physics, to spend too much on costly equipment in geophysics for the purpose of conducting work on some abstruse research problem, for the solution of which there is neither pressing need nor researchers. I recommend the following in the hope of preventing unnecessary expenditure of money by schools and of saving enthusiastic students of geophysics from early disappointment:

The universities and institutions which maintain strong departments of physics and mathematics should offer their students two separate groups of courses in geophysics. The first, or preliminary group, should include the following subjects:

	Lecture Hours	Laboratory Hours	Field Work
Magnetic prospecting..	20	15	3 days
Torsion balance.....	20	30	6 days
Seismic prospecting....	20	30	6 days
Electric prospecting...	20	30	3 days

After a successful completion of at least two courses in this preliminary group, a student would be given the right to join a research group, to study general geophysics, and to do research under supervision of instructors in geophysics. A research student would be advised to carry a heavy schedule in mathematics and physics because no real research work can be carried on without sound training in these fundamentals.

Technical and mining schools should cater to local needs and concentrate on teaching the preliminaries on a somewhat wider scale—that is, by giving more attention to individual training. Large expenditures of money on costly equipment will undoubtedly have an unfavorable effect on securing the proper recognition for geophysics.

V. GAVRILOVICH GABRIEL.

New York City.

J. H. Hieber, until recently with the Balotoc Mining Company, is in charge of development at the Hartwell mine of Benguet Consolidated, Philippine Islands.

Col. O. B. Perry, consulting mining engineer, of San Francisco, was one of six men awarded the honorary degree of doctor of laws, by the University of Indiana, at Bloomington, on April 2, in connection with the ceremony of dedicating the university's new chemistry building. Colonel Perry was for many years consulting engineer to the Guggenheim Exploration Company, of New York.

Hugh W. Sanford, of Knoxville, Tenn., proposes that the United States government take over all gold mines in the country that are now idle because of low grade of ore, and operate them with a staff of unemployed, allowing the men the gold they mine, after deducting charges for food and lodging, and minting the coins. He suggests that the government buy the mines, transport the men thither, allowing them to accept more remunerative work elsewhere as soon as it becomes available. Mr. Sanford goes on to say that the workers should get as wages ". . . all the gold they can produce. The production of these men will not compete with the output of anyone else, being gold, and will therefore cause no further unemployment. The output will require no selling, and will increase the amount of money in circulation, thus helping to bring back good times."

OBITUARY

John S. Cocking, of Calumet, Mich., superintendent of construction for Calumet & Hecla, died recently at the age of 64. He was in the employ of the company for 40 years.

S. W. Farnham, mining engineer with the Goodman Manufacturing Company, died recently in Chicago. Of late years he had taken many trips abroad in the interests of his company.

Walter M. Sanders, mining engineer, of New York City, died recently in a hospital in Montclair, N. J. He was a graduate of Yale University, class of 1899, and was 52 years old.

Edward Allen Taylor, mining engineer, of Salt Lake City, formerly active in the Park City district of Utah, died at Altadena, Calif., recently. Mr. Taylor was born in Salt Lake City in 1872.

Charles W. Sutro, of San Francisco, former stockbroker and president of Katherine Gold Mining, died on April 2, after a long illness. Mr. Sutro, who was 65 years of age, was born in British Columbia, but he had spent most of his life in San Francisco.

Nelson S. Greensfelder, advertising manager of the Hercules Powder Company, died on April 5, at Wilmington, Del. He was born in St. Louis 40 years ago, and was a graduate of the Colorado School of Mines. He is survived by his widow, a son, and his parents.

Fred W. Price, manager of the Rico Argentine Mining Company, died at Salt Lake City on April 2. He was born in Cardiff, Wales, and came to the United States at the age of sixteen. Mr. Price, who was 68 years old, was the discoverer of the Cardiff mine, in Big Cottonwood Canyon, near Salt Lake.

Willard B. Daniels, mineralogist, died suddenly at his home on Staten Island, New York, on March 22. Mr. Daniels, who was 56 years of age, was treasurer and a director of Whittaker, Clarke & Daniels, of New York, importers and exporters of minerals. He had been in the business for more than 40 years, and frequently did consulting work for the United States government.

Ike Thompson died on March 28, at Jedway, Queen Charlotte Islands, British Columbia. He was a pioneer mining man, well known in the western United States and Canada. Mr. Thompson was active in Colorado, Idaho, and Montana before he finally settled in British Columbia. He took part in the Atlin and Klondike rushes. In recent years he served as mining recorder at Jedway.

Charles V. Jenkins, formerly secretary and treasurer of the Nevada Consolidated Copper Company, died in San Francisco on March 17. He was well known in Utah mining districts, having lived for many years in Park City. Mr. Jenkins, who was 64 years of age, was traveling auditor for the Guggenheim interests for some years prior to his appointment as business manager of Nevada Consolidated.

NEWS OF THE INDUSTRY

Grade in Juneau North Orebody About $2.60, Bradley Says

In a general way, a grade of $2.60 per ton, largely in gold, is indicated in the North orebody of Alaska Juneau below the Gold Creek adit level, according to F. W. Bradley, president of the company. Since publication of the company's annual report (in which extended reference is made to development in the North orebody on and below the Gold Creek level—see *E.&M.J.* of March 23), Alaska Juneau has been very active on the New York stock market. Cost of operating, together with pumping, is estimated to indicate a profit of about $1 per ton from the ore at this horizon. At present, ore supplies for the mill at Juneau, Alaska, are principally from the south orebody above the adit level. The grade is about $1.10 per ton and profits have been ranging from 40 to 50c. a ton.

The lateral area indicated in the North orebody is seven times that of the Alaska Treadwell mine, which was operated to a maximum depth of 2,000 ft. below sea level before being flooded. Production of Alaska Treadwell was 16,349,633 short tons, from which $1.99 per ton was recovered. Average grade was about $2.50 per ton. Production of Alaska Juneau in the first quarter of 1931 was $990,500, compared with $792,500 in the same period of 1930.

■

H. W. Gould Options California Low-Grade Gold Property

H. W. Gould & Company has taken an option on the Big Canyon gold property, 5 miles south of Shingle Springs, El Dorado County, Calif., with the possibility of developing a new large-scale low-grade gold mine. Big Canyon is owned by the Hayward Lane Estates. Preliminary examination, now under way, will be followed by unwatering.

According to engineers' reports, the vein material averages nearly 40 ft. wide and the oreshoot is 1,800 ft. long. The mine has been developed to a depth of 800 ft. Grade is estimated to be between $4 and $4.50 per short ton. The mine was opened in 1896 by Hayward and Lane and closed in 1900. At one time it was equipped with a 30-stamp mill and chlorination plant. If the mine develops as expected, the option-holders plan to install a 500-ton mill and flotation plant, cyaniding the concentrate on the ground.

■

Santa Maria Mine Reopens

Operations have been resumed at the Santa Maria de La Paz silver mine, in the Matehuala district, San Luis Potosi, Mexico, which shut down in February. The resumption has been made at the request of the government, which desires to avert unemployment. Negotiations for the transfer of the property to another company are reported to be under consideration.

Nevada Cons. Abandoning Wedge Shaft at Ely Mines—Adds to Chino Reserves

Nevada Consolidated is abandoning the Wedge shaft, sunk to provide an outlet for underground operations at its Nevada mines, in the Ely mining district of Nevada, according to statements made in the company's annual report for 1930. Although put down in 1925 and 1926, this shaft has never actually been used for regular mining operations, as sufficient extensions to the ore amenable to mining by power shovel have been found to maintain production. The new Monitor shaft, 1,300 ft. north of the Wedge shaft, will be used for underground operations. At the Chino mines, the annual report reveals, drilling south of the Sierra orebody has added a total of about 60,000,000 short tons, making total reserves at all mines more than 300,000,000 tons.

According to the report, substantial savings made possible by electrification of shovel operations have rendered economically possible extension of the Nevada pit program beyond what appeared some years ago to be its probable limits. The Wedge shaft is, therefore, being abandoned as the main hoisting shaft, the equipment having already been moved to the Monitor. The Wedge will be retained as an auxiliary for ventilation and general utility purposes. Electrification of shovel equipment at the open pit will be started this year and continued as rapidly as economic conditions permit.

At Chino mines, near Hurley, N. M., development of a large additional tonnage of ore that can be mined by shovels has rendered unnecessary any further preparation for underground mining. All work for this purpose, both surface and underground, has been stopped. Two more steam shovels will be electrified during 1931, making a total of ten electric shovels. Concentrator remodeling has been completed and the plant has an effective capacity of 15,000 short tons daily.

Underground operations at Ray have been confined to the No. 2 shaft area and are now on a one-shift basis. Exploration has been limited to the development of ground northwest of the present workings. About 1,000,000 tons of ore above the No. 3 level has been proved, existence of which had heretofore been questionable. No. 2 headframe has been rebuilt. No change has been made at the mill.

Comparative cost data for 1929 and 1930 at the mines and mills follow:

	1929,	1930
Chino	31.38	33.40
Nevada pit	24.15	28.15
Nevada Ruth Mine	95.00	93.6
Ray	82.75	67.99
Milling costs, per ton:		
Chino	59.00	54.53
Nevada	42.10	45.77
Ray	39.45	38.16
Copper, per lb	9.33	10.85

Copper per-pound costs include depreciation and precious-metal credits, but not federal taxes.

At the smelter at McGill, Nev., the new Cottrell plant has shown an operating and maintenance cost two-thirds lower than that of the old flue system. No important changes have been made at the smelter or concentrator, but research on milling operations indicates that some improvement can be made in recovery and concentrate grade by minor rearrangements.

■

U.V.X. Cuts Vein at Vulture

Sinking of the three-compartment shaft that United Verde Extension is putting down at the old Vulture gold mine, near Wickenburg, Ariz., cut the vein at a depth of 509 ft. on April 2. Where cut, the vein did not show ore, according to a report from Tucson. U.V.X. is now developing the property under option. Water flow, encountered at about 450 ft., has hampered recent shaft sinking. The interest which this work has aroused in adjoining gold properties is reflected in the recent sale of the Blinkenstaf group of ten claims to a San Francisco syndicate, headed by Edward Johnson. Diamond drilling has been started, although a shallow shaft had been sunk and some drifting done by former owners.

United Verde Extension has taken a lease and bond on the Lorraine copper property, held by Pinal Copper, near Ray, Ariz. Development has been concentrated on two veins, from one of which two shipments of high-grade copper ore have been made in the last two years. The property covers 700 acres.

■

New Ore Found at Mascot

Ore has been found at four "new and separate spots" on territory adjoining the Mascot and Jarnagin mines of American Zinc, Lead & Smelting, near Knoxville, Tenn. This territory is under option to the company. Because of current conditions, exploration has been curtailed, but, with improvement, work will be resumed to further prove the extent of these discoveries, Howard I. Young, president of the company, states. Because of the "comfortable ore reserve position," little new work was done at the Mascot.

American Zinc has sold the Douthat mine, near Cardin, Okla., after all ore that could be mined by the company was exhausted. The High Five property, near Waco, Mo., in which it had a 50 per cent operating interest, also has been sold. In addition, American Zinc disposed of a large acreage in Newton County, Mo. Net loss resulting from sale of capital assets was $947,009, deducted from capital surplus.

Four Tunnels at Hoover Dam Will Be 1 Mile Long Each

Work will start immediately upon driving of four diversion tunnels at the Hoover Dam project, on the Colorado River, near Las Vegas, Nev. These tunnels will carry the waters of the Colorado River through the canyon walls around the dam site while the dam is being built. Each tunnel will be 57 ft. in diameter and 1 mile long. Completion is expected within eighteen months by Six Companies, Inc., the contractors who will build the dam. A total of 1,563,000 cu.yd. of rock will be excavated in driving them.

Air-compressor and rock-drilling equipment has been ordered from Ingersoll-Rand for this project. The stationary air plant will consist of a battery of direct-connected, electric-driven compressors with a combined output of 25,000 cu.ft. per minute. It is expected that the work will be carried on from one heading in each tunnel only, as the time allowance is sufficient. The exact method of drilling to be followed will probably be determined by rock conditions as found after a start has been made. Because of the unusual diameter of the tunnel, the work will almost come under the heading of quarrying rather than tunneling. All excavated material from the tunnels will have to be hoisted up the walls of the Grand Canyon for disposal, as no space is available at the river bottom.

■

Nichols Copper Obtains Control of Freeman Roasting Process

Nichols Copper, subsidiary of Phelps Dodge, has acquired a controlling interest in the Freeman flash process for roasting sulphide ores. Horace Freeman has recently worked out the details of the process in an experimental plant at Shawinigan Falls, Que., and patents have been obtained in Canada. Particular success has been attained in the application of the process to the roasting of pyrite to make sulphur dioxide gas for use in the paper industry. Mr. Freeman last month was awarded a gold medal by the Canadian Institute of Mining and Metallurgy.

The Freeman patents are owned by the Sulphide Research Corporation, Ltd., a Canadian corporation. Nichols Engineering & Research Corporation, incorporated under the laws of Delaware, has been formed by the Nichols interests to consolidate all their Herreshoff furnace patents with all of the Freeman patents. The merger was effected by Nichols transferring its Herreshoff furnace department to the new corporation in return for common stock, while Freeman and associates exchanged all of the capital stock of Sulphide Research for common stock in Nichols Engineering & Research. Nichols undertakes to purchase preferred stock of the new corporation in an amount sufficient to furnish necessary working capital.

The board of directors of Nichols Engineering & Research, containing representatives of both interests, follows: One-year term: N. C. Timmins, D. R. Nichols, J. B. Beaty. Two-year term: Louis S. Cates, A. T. Thomson, Andrew A. MacKay. Three-year term: C. W. Nichols, H. J. Hartley, Horace Freeman. The officers of the new corporation will be as follows: C. W. Nichols, chairman of the board; H. J. Hartley, president; Horace Freeman, vice-president; A. T. Thomson, secretary and treasurer; and J. B. Beaty, assistant secretary.

Sulphide Research Corporation will continue to carry on all development work, including installation of equipment, for the utilization of the Freeman process in Canada, the board of directors of that company being increased from five to seven by the addition of C. W. Nichols and H. J. Hartley.

The Royal Bank of Canada, in its bulletin for 1931, commented favorably on the importance of the Freeman process for the newsprint industry alone, stating that the annual cost of sulphur was $4,000,000 and estimating that $1,652,000 of this could be saved by the use of pyrite.

■

Zinc Institute Meets in St. Louis April 20-22

Speakers at the thirteenth annual meeting of the American Zinc Institute, to be held at the Statler Hotel, in St. Louis, Mo., are as follows:

April 20—Afternoon session: E. J. Mehren, vice-president, McGraw-Hill Publishing Company, "A Platform for American Business."

April 21—Morning session: J. D. Conover, secretary, American Zinc Institute, "The Institute's Galvanizing Program"; G. C. Bartells, technical assistant, American Zinc Institute, "Will the Farmer Buy Heavily Galvanized Sheets?"; Larry Miller, vice-president, Granite City Steel Company, "What the Steel Roller Thinks of Heavily Galvanized Sheets"; Roy DeStaebler, sales manager, Beck & Corbitt Company, "What the Jobber Thinks of Heavily Galvanized Sheets." Afternoon session: S. H. Levison, American Smelting & Refining, "Zinc Smelting in Mexico"; Foster S. Naething, Evans-Wallower Zinc Company, "Evans-Wallower Electrolytic Zinc Plant"; Elmer Isern, Commerce Mining & Royalty, "Centralized Milling in the Tri-State District"; H. A. Nelson, New Jersey Zinc Company, "Zinc Dust as a Protective Coating." John C. Howell, consulting economist, will speak on "The Business Outlook" at the annual banquet, to be held in the evening.

April 22—Morning session: C. S. Trewin, secretary, Subcommittee on Embrittlement, American Society for Testing Materials, "What Is the So-Called 'Embrittlement' of Galvanized Structural Steel?"; Prof. H. R. Hanley, Missouri School of Mines, "Status of Zinc Processes"; R. S. Dean, U. S. Bureau of Mines, "Reduction of Zinc Ore by Methane Gas"; R. B. Caples, Anaconda Copper, "Recovery of Zinc From Lead Blast Furnace Slags"; H. A. Buehler, for St. Louis Smelting & Refining, "Zinc Developments at Pioche, Nev."

■

Farmers Appeal From Fume Decision

The decision of the International Joint Commission, awarding $350,000 to farmers in Northport County, Wash., for crop damage from smelter fumes coming from the Trail plant of Consolidated Mining & Smelting, in British Columbia, has evidently not satisfied the victors. They have appealed to Secretary of State Stimson, of the United States, who heard their claims on March 31. Originally, more than $3,000,000 in damages was asked.

Gold Placer Strike, Said to Be Rich, Made in Sinaloa

A strike of placer gold that has attracted unemployed peons from all sections of northwestern Mexico has been made near San Ignacio, Sinaloa, Mexico. Newspaper reports are confused, but all of the accounts agree in describing the strike as being unusually rich, the exodus of peons unusually large, and the place as being at El Tambor, in northeastern Sinaloa, 100 miles from Mazatlan. Because of the strike, Mexican authorities have imposed the strictest border regulations, one Tucson editor states, and are closely guarding the district itself. All foreigners, particularly Americans, are being excluded.

The Tambor district is in the hands of the most powerful politicians of the State of Sinaloa. As the gold is largely placer, little capital will be needed for its recovery and the full benefits of the discovery can be retained for Mexicans. Prospectors returning from the strike display some wonderful nuggets. Reports are that within the last week or two, several hundred thousand dollars in gold has been shipped out and that the mother lode has been discovered.

The first newspaper report to appear, an Associated Press dispatch from San Ignacio, stated on March 20 that Guillermo Laveaga, once a Mexican senator, had returned to San Ignacio. He declared that he had spent six months on the field and that 1,200 lb. of gold—worth $350,000—had been recovered, largely from the El Tambor River. At that time he stated the number of claims filed were 1,600 and that the field had been discovered by bandits. The nephew of one of these revealed it last fall to an old woman in San Ignacio. After Sr. Laveaga's statement, a rush immediately started. A subsequent A.P. dispatch, on March 31, stated that Manuel Llantada, self-styled co-discoverer of the vein, had found nuggets as "big as onions." Two carloads of ore taken from his claim, according to government assays, carried $149,000 in gold to the ton. Hundreds of peons were on the field, Llantada said, and 33,000 clams had been filed in all. Farm production in the neighborhood had ceased, he added, and the price of food on the field was soaring. On April 4, another A.P. dispatch gave the number of men on the field as 1,600. A controversy has arisen between Cosala and San Ignacio as to which town exercises jurisdiction over the district.

Engineering and Mining Journal has not yet seen a report on the field by a known mining engineer or by the Mexican government—neither has there been any official statement of production.

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Propose New Mexican Labor Code

A new Mexican labor code has been proposed, with the sponsorship of Aaron Saenz, Secretary of Industry, Commerce, and Labor. It replaces the code introduced in the fall of 1929, which aroused widespread opposition and was never adopted. The new code, Sr. Saenz states, is designed to increase Mexican industrial production, and "seeks to understand the viewpoint of capital." Among the most important provisions are one for a minimum wage, based on the cost of living, and another for conciliation in the settlement of industrial disputes.

Con. Coppermines Development Successful During 1930

Churn drilling and development in Emma-Nevada ground at its properties in the Ely district, Nev., added about 5,000,000 short tons to ore reserves of Consolidated Coppermines during 1930, according to J. B. Haffner, general manager. On Jan. 1, 1931, reserves were 35,000,000 tons, averaging 1.1 per cent copper. Other favorable developments of low-grade ore were made in the Minnesota territory, but an estimate of tonnage is not yet possible. In the Alpha shaft, high-grade reserves were increased.

Detailed geologic study of the Minnesota area (from which shallow, secondary ores have been produced in the past) indicated the possibility of extensive primary mineralization at considerable depth. Two churn-drill holes have been put down and have cut a primary orebody varying in thickness from 185 to 305 ft. One hole cut a higher-lying secondary orebody 150 ft. thick.

At the Alpha mine, the 1,400 level has proved a larger tonnage in the so-called old-ore zone than the 1,300 level. To date, ore has been developed for a length of 250 ft. on the 1,400 level, with an average width of 25 ft. and a grade of 8 per cent copper. Limits have not yet been reached. In 1930, Alpha shipped ore averaging 7.7 per cent copper, compared with 5.4 per cent in 1929. Crosscutting on the 1,600 and 1,800 levels has drained the overlying ground, preliminary to development.

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Work at Depth in Ahmeek Shows Drop in Grade

Development at depth in the Ahmeek unit, of Calumet & Hecla, indicates that the grade of the ore is declining as the operations attain greater depth, according to the annual report. The Ahmeek unit and the adjoining Allouez, Centennial, and Kearsarge mines are at the northern end of the company's producing properties in the Michigan copper district. They provide a large proportion of the company's total ore output—1,193,704 short tons in 1930, or about 40 per cent. Grade in 1930 averaged 26.9 lb. of copper per ton, compared with 29 lb. in 1929.

Nos. 3 and 4 Ahmeek shafts were sunk 726 ft. in 1930. The ground opened by 6,013 ft. of new drifts from these shafts is of lower grade than that opened in previous years and cannot be mined at a profit under present copper prices. Drifting south from No. 2 shaft into Allouez territory on Nos. 34 to 37 levels opened some low-grade ore. Work done both north and south of the shaft in Ahmeek ground, between the Nos. 38 and 40 levels, also showed low-grade material. Drifting from No. 2 Centennial shaft, now down to 222 ft. below the No. 41 level, opened good-grade ore.

The copper content of ore from the Calumet & Hecla conglomerate unit showed an increase in grade from 42 lb. per ton in 1929 to 48.7 lb. per ton in 1930. Nevertheless, drifting at depth from "A" and "C" shaft indicated that mineralization lacks continuity and is generally of low grade. Some work was done as deep as the No. 96 level from the "A" shaft.

In the Osceola amygdaloid unit, the third of the C. & H. producing groups, the No. 16 shaft has been closed down, because of its limited production. The grade of ore developed by a total of 23,253 ft. of drifting was about the same as that opened in 1929.

Average recovery, per ton of tailing treated in the reclamation plants at Lake Linden, was 11.55 lb. of copper. This compares with an average of 10.85 lb. since operations started. Improved recovery has been achieved despite a drop in grade. The assay of all material treated in 1930 averaged 0.665 per cent copper, compared with 0.680 per cent, the average treated since starting. Actual production was considerably less than in 1929, the result of an interruption to operations in midsummer and suspension last November. Both dredges are still closed down, pending improvement in the copper market.

Cost of producing mine copper dropped from 11.43c. a pound in 1930 to 10.56c. in 1929. Reclamation-plant costs increased from 5.62c. a pound to 6.71c.

Falconbridge Reserves Rise

Reserves of nickel-copper ore at the Falconbridge Nickel property, near Sudbury, Ont., were estimated at 2,279,453 short tons on Dec. 31, 1930, compared with 1,367,000 tons on Oct. 31, 1929. However, the reserves at the end of 1930 include 669,710 tons below the 1,000 level, calculated entirely on diamond-drilling results. The grade of the ore is put at 2.47 per cent nickel and 1.08 per cent copper. Present smelter capacity of 300 tons daily can be increased about 40 per cent with a minimum of delay, E. Craig, superintendent, states in the annual report. Refinery capacity, now 3,000 short tons of nickel annually, can be increased to 4,000 tons at a moderate cost, though with several months' delay in time. The refinery is in Norway and treats about 750 tons annually for Norwegian producers.

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Miami Costs Reduced to 77c. a Ton; Grade Drops but Extraction Is Up

AT THE end of 1930, Miami Copper's operating costs had been reduced to 77c. a short ton of ore, F. W. Maclennan, general manager of the company, at Miami, Ariz., reveals in the annual report. The recovery of copper, per ton of ore handled, was down to 10.5 lb., however. This recovery does not reflect a drop in efficiency, for extraction during 1930 improved to nearly 79 per cent of the copper content. It is an indication of the extremely low grade of ore handled—probably the lowest-grade copper ore mined anywhere and certainly the lowest grade in Miami's operating history. In 1930, the 6,124,993 tons of ore sent to the mill assayed 0.716 per cent copper.

Underground development at the Miami mine is now centered largely on preparation of stopes on the 720 and 1,000 haulage levels. Total footage driven in 1930 was 77,430 ft., of which 32,422 was on development account and the remainder on stoping account. In 1929, total footage was 131,360 ft., of which 96,455 was development. In addition, 3,422 ft. of diamond drilling was completed in 1930. Of this, 2,372 ft. was drilled from the 520 level for the purpose of determining the western boundry of the orebody and 1,050 ft. was from the 850 level, to determine the bottom of the ore to be served by the 1,000 haulage level.

A new inclined shaft, No. 6, was started from the surface in August, 1930. It will connect with the 620 grizzly level and afford better ventilation for the lower levels of the mine. No. 4 shaft has been deepened to the 850 level.

The following tabulation indicates the improvements in extraction and the remarkable reductions in cost that were achieved in the course of one year at this property:

Per Ton:	1929	1930
Mining costs, cents	42.3	39.6
Milling costs, cents	35.9	29.7
General costs, cents	15.2	13.9
Total costs, cents	93.4	83.2
Assay, lb. of copper	16.6	14.3
Recovery, lb. of copper	11.72	10.96
Concentrate, per cent copper	36.24	34.13
Mill extraction, per cent	72.8	78.9

In the last quarter of 1930 Miami reduced its operating costs, per ton of ore handled, to 77.1c., although the amount of copper recovered dropped further to 10.56 lb. per ton. The cost of producing copper to concentrates on board cars at Miami was reduced in the last quarter to 7.3c. a pound, compared with 7.6c. for the entire year, and 7.97c. a pound in 1929.

No new ore was developed during 1929, the ore reserves therefore being reduced by the tonnage mined. On Jan. 1, they were estimated at 84,276,787 tons, averaging 0.88 per cent copper, of sulphide ore, and 7,000,000 tons, averaging 1.83 per cent copper, of mixed ore. A considerable amount of laboratory work has been carried on during the year with a view to developing a process for this mixed ore which would not involve a large outlay of capital. Results of this experimental work have been encouraging so far, Mr. Maclennan states. Large-scale tests will be undertaken during the coming year with the object of confirming these results.

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Patiño on One-Shift Basis

Patiño Mines is now operating on a basis of one shift daily at the mines, and two shifts daily at the mill, the annual report of this Bolivian company reveals. Production has been stabilized at about 1,300 long tons of fine tin monthly. In 1930, total tin content of concentrate produced was 17,358 metric tons, compared with 21,108 tons in the preceding year.

Completion of a program of expansion of compressed-air and power plants has been delayed. Total new workings in 1930 measured 21,521 m., compared with 25,346 m. in 1929. At the close of the year, reserves were estimated at 1,876,473 metric tons, averaging 4.11 per cent tin. At the close of 1929, reserves were 1,964,864 tons, averaging 4.2 per cent tin. The cost of fine tin, delivered in London, excluding depreciation, was £118 9s. per long ton in 1930. In 1929 it was £134 15s.

New Ore Prolongs Life of Humboldt Mine at Morenci—May Delay Clay Program

Continued curtailed production at its Morenci unit, in southeastern Arizona, together with ore extensions in the Humboldt area, will make it unnecessary for Phelps Dodge to consider large capital expenditures in connection with the Clay orebody, at Morenci, at as early a date as originally anticipated, according to P. G. Beckett, general manager, in the annual report. Last year Phelps Dodge announced that it had proved 245,000,000 short tons of low-grade ore in the Clay orebody, and that production on a scale of 25,000 tons of ore daily (about 80,000 tons of copper annually) would start probably in 1934.

During 1930 construction of pilot plants for testing Clay ore by leaching and flotation methods was completed. During 1931, representative tests will be run, taking the same feed of representative low-grade ore. Additional diamond drilling has established the outlines of the orebody more definitely.

New ore has been developed in the Humboldt orebody at Morenci below 34 stope on No. 5 level. Sinking of No. 6 ore-hoisting shaft to permit driving of a deeper haulage level has been completed. The shaft is operating satisfactorily with the new hoist, headframe, and 7-ton skips. Two underground service shafts to accommodate lower lift levels have been driven. Remodeling of the Humboldt concentrator has been completed, and its capacity is now 5,000 tons of ore daily. At the Clifton smelter, production of anodes has been discontinued.

At the Copper Queen unit of the company, operations in the higher-grade limestone mines are now centered at the Holbrook and Sacramento shafts, the other units having been closed in the interests of economy. Daily production of limestone ore is about one-third of what it was a year ago. Known orebodies have been satisfactorily extended, but no new ore was found in the Warren shaft, now closed, during 1930. In the porphyry mines, development work has been discontinued in the North Block of the East orebody, which was being prepared for production. Six additional glory-hole raises have been put up at the Sacramento pit, making twelve in all. Costs have been appreciably reduced in this glory-holing operation.

During 1930, the Southeast Extension orebody produced the largest tonnage in its history—560,871 tons, about equally divided between the 850 and 950 levels. The 850 level is now exhausted. Additional ore of good grade has been developed on the 1,000 level, extending the ore outlines at that point. A new haulage level has been started from the Sacramento shaft at 1,200 ft. A drawing-off level has been established at 1,100 ft., giving a total height of ore of 150 ft. This level will produce more than 4,000,000 short tons.

At the concentrator, addition of two ball mills in the secondary grinding units in the No. 3 section has increased capacity of the section by about 70 per cent. Total mill capacity evidently can be increased by this amount at a moderate cost, if desired. At present it is about 4,000 tons daily. Remodeling of the Copper Queen smelter has been satisfactorily completed.

Development of the Nos. 19, 20, and 21 levels at the Moctezuma Copper mine, near Nacozari, Sonora, has been retarded by the comparatively large flow of water encountered. An additional 400-g.p.m. triplex pump has been installed. Although new development footage driven in 1930 was only three-quarters of that in 1929, sufficient new ore was found to maintain ore reserves with little fluctuation in grade. In the Guadalupe shaft, the 2,150-level ore and waste pockets have been completed. Timbering of the New Pilares shaft below the 1,800 level is now under way. Metallurgical results at the concentrator were improved. The only new equipment installed was a 4-ft. Symons cone crusher.

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Ohio Plans Adding Flotation Units to Tailing Plant

Addition of flotation units to the leaching plant erected last year at the Ohio Copper property, Bingham, Utah, for re-treatment of tailing is planned. This will enable treatment of slime from leaching section, now erected, and also of material deeper in the tailing dump, where oxidized content is small. The leaching section was built for about $25,000, with an estimated capacity of 3,000 short tons daily. Extremely cold weather made operation impossible after the plant was completed on Oct. 21. Production will start this spring as soon as weather permits.

About 4,000,000 tons of tailing is available, averaging about 0.45 per cent copper. In the outer portion of the dump, about half the copper is soluble in discard water from the precipitation plant in the Mascotte tunnel. About 80 per cent of this soluble copper can be recovered at a cost of 16c. per ton of tailing (or 5c. a pound of recovered copper) by the leaching unit, consisting of hydraulicking apparatus, settling ponds, and precipitation boxes. Experiments proved that recovery can be increased from 3 to 6 lb. per ton by including flotation. The leaching section alone is expected to yield $15,000 monthly operating profit, with copper at 10c. a pound. Enlargement of the pilot flotation mill can be undertaken with this surplus, and at least the slime from the leaching unit can be treated by flotation. An operating profit of at least $150,000 annually for the next seven or eight years is expected by I. B. Joralemon, consulting engineer.

Early in 1931 the campaign of underground development at the mine was concluded. All three of the main ore-bearing limestone beds of the Bingham district were explored between the Mascotte and 700 levels. A few copper-gold oreshoots were found in replacement bodies of iron pyrite and 1,300 tons of ore was shipped. A larger tonnage of leaner material, remaining in the mine, cannot be treated profitably at present. Exploration at further depth, Mr. Joralemon believes, might yield higher grade of ore. He recommends shaft sinking to the 1,200 level and development of all three limestone beds at that depth. About $100,000 would be required for this work.

Slime Unit Raises Extraction in Inspiration Leaching Plant

During 1930, a further improvement in the metallurgical efficiency of the leaching plant of Inspiration Consolidated, at Inspiration, Ariz., resulted from the use of the new slime-leaching plant. As a result of treating higher-grade ore, suspending production in the flotation concentrator, and improvements in metallurgical operations, costs per pound of copper produced dropped from 11.13c. in 1929 to 9.67c. in 1930. These costs include depreciation but do not include depletion or federal taxes.

Previous to 1929, Inspiration did not remove minus-200-mesh material from ores delivered to its leaching plant. When this classification was introduced in 1929, the leaching extraction improved to 89.1 per cent of the copper content (having been 80.4 per cent in 1928). The slime removed in 1929 was diverted to the concentrator for flotation treatment and was then stored for treatment in the slime-leaching plant. This unit was brought into operation in May, 1930. Since it started production, the total copper extraction from leaching-plant ore has been 90.7 per cent. In addition to improving extraction, desliming of the ore has lowered acid consumption.

At the electrolytic cell house, the spacing of the electrodes in the commercial section has been changed from 4½- to 4-in. centers. This change has increased cell-house capacity from about 160,000 lb. to about 200,000 lb. daily. To take care of increased capacity, four additional starting sheet tanks have been installed.

Underground, a total of 61,191 ft. of new workings was driven in 1930, compared with 135,890 ft. in 1929. In addition, 1,411 ft. of diamond drilling, 140 ft. of churn drilling, and 95 ft. of shaft sinking was accomplished in 1930. The 3,041,854 short tons of ore mined were extracted at an average rate of 15.064 tons per man-shift, compared with 11.313 tons in 1929.

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Christmas Drilling for Ore at Depth in No. 4 Shaft

Diamond drilling from the 500 level of No. 4 shaft at Christmas Copper, Christmas, Ariz., is being undertaken to determine at what depth ore occurs east of the Christmas fault, according to F. P. Knight, president of Iron Cap Copper, the controlling company. The area favorable for ore occurrence at the property is divided nearly in half by the fault. Subsequent to the formation of the ore, the east half dropped probably 1,500 ft., Mr. Knight states. No. 4 shaft is situated in this eastern area. Although the first drill hole had not reached limestone at the time of the preparation of the annual report (March 17), 1,200 ft. of mineralization has been disclosed by the hole and the shaft.

The drift to connect the No. 4 shaft with the No. 3 shaft, which is now used for ore hoisting, has been stopped. It is on the 500 level of No. 4, but corresponds to the 1,000 level of No. 3. Despite extraction of 216,604 short tons of ore during 1930, development increased reserves. Nearly all the ore produced during the year came from the upper horizon at No. 3 shaft. To date, development on the lower horizon at the 800 level has disclosed promising conditions, both as to ore and grade.

BRIEF NOTES

Altar Mining & Exploration. The company has purchased a 100-ton smelter at Noria, Sonora, Mexico, and has engaged Jackson Brothers, of Wickenburg, Ariz., to dismantle the plant and re-erect it at its property in the El Plomo-Altar district of Sonora. For the present, the company states that it will concentrate on production of gold and silver, using only enough copper ore to act as a collector for the precious metals.

Ashley Gold. About 100 men are now employed at the property in Bannockburn Township, Ont. Sinking of the three-compartment shaft has reached a depth of 50 ft. Diamond drilling is being continued to explore the extension of the known ore sections of the Ashley vein, and a second diamond drill is in operation on the Garvey discovery. According to the annual report of Mining Corporation of Canada, controlling factor in the company, electric power at reasonable cost will eventually be available at points either 9 miles to the east or 28 miles to the west.

California-Ahumada Mining. Production from the La Gloria mine, near Estacion Alcaparra, Chihuahua, Mexico, has virtually ceased, and development is being cut to the minimum necessary to comply with terms of the company's lease. During 1930, the orebodies failed to develop as expected. However, work to the south of "D" orebody opened up an iron deposit containing some lead and silver. On the 80-m. level it is about 70 ft. wide and more than 150 ft. long. The shaft has been deepened to 110 m., and development at that depth shows ore, in some places, running as high as 2 per cent copper and 35 oz. of silver to the short ton. In 1930, the average grade of ore shipped was 22.4 per cent lead and 42 oz. of silver to the ton.

Central Eureka. The company has drifted a distance of 370 ft. in ore on the 2,100 level of its Old Eureka property, at Sutter Creek, Calif. The ore is about 5 ft. wide and of milling grade. On the 2,000 and 2,300 levels, ore has been cut. Production is about 3,750 tons of ore a month.

Dawson Gold Mines. Drifting north along the Dyke vein, at the property near Verona, B. C., has followed high-grade ore for a distance of 40 ft. The company plans to sink a winze on this vein and then to extend a deeper tunnel to cut it, if the winze shows continuation at depth. The vein is only 2 ft. wide. A shipment of 95 short tons, made from the property, showed a gross return of $378 in gold per ton. Sampling of the new ore developed indicates at least as high a grade, according to W. G. Norrie, consulting engineer.

Engels Copper. Diamond drilling indicated ore running about 1 to 4.5 per cent copper at a depth of 150 ft. below the No. 16 level of the Engels mine, in Plumas County, Calif., before operations were suspended at the property in the summer of 1930, W. I. Nelson, superintendent, states in the annual report of Engels Copper. Ore reserves blocked out, however, are only 138,105 short tons. Other development failed to disclose new ore. At the Calaveras mine, Copperopolis, Calif., drifting had shown ore averaging 5 per cent copper over a width of 10 ft. and for a length of 15 ft. on the 675 level, near the Keystone shaft, when operations were suspended in July. This ore is expected to continue for some distance. Installation of flotation equipment resulted in improved extraction from the concentrator, which handled more than 200 tons daily of an average grade of 2.5 per cent copper.

Eureka Standard. During 1930, development work at this property, in the East Tintic district of Utah, increased ore reserves despite the extraction of 28,459 short tons of ore. The average grade of this ore was 1.07 oz. of gold and 10.4 oz. of silver per ton, in addition to a small, but recoverable, lead and copper content. In 1929, the grade was 0.8 oz. of gold and 10.5 oz. of silver per ton. Enlarged operations at the property aided considerably in mitigating the unemployment situation in the district, caused by low prices for lead and silver. Development work done in 1930 included 3,117 ft. of drifts, 1,930 ft. of raises, 84 ft. of shaft sinking, and 141 ft. of winze sinking.

Hanlon's Reward. According to T. B. Crosbie, of Melbourne, a director in the company, three shafts have been sunk to a depth of 100 ft. at the property on the Jervois Range, Central Australia, and development has been undertaken at that depth. Two of the shafts develop copper deposits and the other shaft has disclosed a silver-lead body. One shaft struck water at 95 ft. The cost of this work has been £10,000. The company hopes to interest English and American capital. Cost of constructing a light railroad from the field to connect with the railhead at Dajarra, southwest of Cloncurry, Queensland, is estimated at £200,000. The property is 180 miles northeast of Alice Springs.

Hedley Gold. Reserves of ore at the company's Nickel Plate mine, near Hedley, B. C., have been nearly exhausted. The company has adopted a plan of diamond drilling from the surface of the Climax claim. About $52,000 is available. If this work fails to find ore, the property will either have to be closed or operated under lease. Drilling has indicated that the main ore lens does not extend in depth below the 1,500 level.

Hudson Bay Mining & Smelting. By the end of 1930, most, but not all, of the major adjustments for the efficient operation of the mine, mill, and metallurgical plants at Flin Flon, Man., were completed, according to a report by R. E. Phelan, general manager. Underground, three stopes have been opened up to permit output of 1,000 short tons daily, in addition to development. The main approach to the open pit has been completed. Open-pit ore mined in 1930 was from the oxidized top of the orebody, which must be removed first. Recovery from this oxide ore is naturally lower than from the sulphide ore, as was demonstrated by the better results obtained from operating on the underground sulphides. The mill was raised to a daily tonnage of 2,700 by the close of 1930. Operation of the zinc plant since November has been satisfactory and continuous. Some initial difficulties were experienced with the copper smelter, but by December these had been overcome. No data on actual metal production, costs, or ore reserves are contained in the report, their omission probably being because of the short period of regular operation.

Isle Royale. After the available stoping ground in No. 2 shaft has been mined, this shaft will again be closed. Only 11 per cent of the drifting on the vein through this shaft, at the property near Houghton, Mich., showed ore in 1930. In No. 4 shaft, 42 per cent, and in No. 5 shaft, 60 per cent, of the new drifting in 1930 opened ore. Grade of ore has been dropping steadily. In 1930, only 20.89 lb. was recovered per ton of ore treated, compared with 24.49 lb. per ton in 1927. Cost of production was cut from 14c. in 1929 to 12.9c. in 1930.

International Nickel. Custodis Canadian has been awarded the contract for a new 350-ft. reinforced concrete chimney to be erected at the Orford separation-process plant at Copper Cliff, on which work is now under way. The inside diameter of the chimney will be about 26 ft. Concrete was poured about April 10 and the chimney is expected to be completed within five months.

Katherine Gold. The property, 35 miles north of Kingman, Ariz., is offered for sale on a competitive basis. Bids will be accepted up to April 21 and must be sent to Room 507, Luhrs Building, Phoenix, Ariz. Katherine is equipped with a 200-ton cyanide mill, which is included in the sale. Ore is comparatively low grade.

Landmark Gold. A syndicate, reported to include Thomas F. Cole, of Pasadena, Calif., has acquired control of the mine, near Tolicha, Nev. Recently, leasers have shipped three carloads of ore that returned $35 in gold per ton from the property. Drifting on ore is being continued by the new owners.

Mayflower Mines. At its property in the East Park City district, Utah, the company is drifting and raising on a fissure cut about 3,000 ft. from the portal of the tunnel. Streaks of sulphide are carried in the fissure. The management expects to cut the limestone formation within a short distance of the tunnel and, at a point 125 ft. from the tunnel in the drift, the intersection of two fissures in the quartzite.

Mexican Premier. The company has obtained ten additional groups of claims, adjoining its properties near Mazatlan, Sinaloa, Mexico, according to a statement at the annual meeting on March 25. Most of the properties carry silver-gold ores and have been worked for several years. Although actual production of concentrate increased in 1930 over 1929 (4,004 metric tons, compared with 3,913 in 1929), the company operated at a loss because of low metal prices. Operations are being continued.

Mineral Veins Coalition. Valuable cerrusite crystals are being mined from the carbonate orebody of the West Toledo mine, near Alta, Utah. Several consignments of these crystals have been shipped to the Smithsonian Institute, the University of Utah, and Leland Stanford University for exhibition and metallurgical test purposes. The crystals carry 79 per cent lead and contain practically no impurities save calcium and mechanically included matter.

Mohawk Mining. In sinking the "E" shaft at the Michigan mine, Ontonagon County, Mich., the company has been following the course of a lode from which several small masses of copper have been recovered in the last 200 ft., according to Theodore Dengler, general manager. The ground is much firmer than in the upper levels and the shaft will be continued a few feet further to an inclined

(Continued on page 340)

EXCLUSIVE

CABLES

FROM CORRESPONDENTS IN THE PRINCIPAL MINING CENTERS

Wiluna Gold in Production; Isa Water Flow Lessens

Melbourne, March 28, 1931.—The roasters at the Wiluna Gold cyanide mill, near Meekatharra, Western Australia, were put into operation on March 10. All sections of the property, both underground and on the surface, are now said to be in satisfactory operation, with the tonnage being gradually increased to the capacity of 40,000 long tons monthly. The Victor Leggo Company, chemical manufacturers of Melbourne, have built an arsenic recovery unit at the property. They will utilize the entire production of residue from the mill.

. . . . The water flow underground at the Mount Isa mine has greatly decreased. Production from this Queensland lead-zinc property is now expected to start by the middle of April, when the crushing unit will be put into operation. By June the company hopes to be treating 10,500 long tons of ore a week. At first the ore handled will be largely zinc-free carbonates and sulphides with a low zinc content. As no provision has yet been made for handling the zinc concentrate, it is possible that zinc will not be recovered when operations first start. Silver-lead bullion will be shipped to England to market. Mount Isa has acquired 140 acres at Cloncurry, to the south, from which it expects to produce ore suitable for fluxing purposes.

. . . . The British government is still holding 127,000 long tons of Australian zinc concentrate that it acquired under the terms of the Board of Trade contract, which expired June 30, 1930. Before final realization is achieved, it is expected that an additional loss of £246,900 will be realized.

. . . . Gold production from the State of Victoria in February was 2,377 fine oz., an increase of 1,024 oz. over production in February, 1930. In January, Victoria gold production was 2,079 oz., an increase of 1,127 oz. over January, 1930. These increases, although they are, of course, small when viewed from the standpoint of total world production, are significant, as they come nearly entirely from individual prospectors. Many small nuggets, up to 10 oz. in weight, are being found in old fields by unemployed men.

. . . . A serious fire at the Mount Morgan mine, Rockhampton, Queensland, has destroyed the ore bins and surface plant. Efforts have been made several times to reopen this well-known producer, particularly in view of the high gold content of its copper ore. A small, though regular, production has been obtained during the last year through precipitation of copper from the mine waters. Tests by the American Cyanamid Company had indicated that the ore could be treated by flotation. What effect the fire will have on the plans of the company is not known.

. . . . Figures for production of metals in New South Wales have just been made available. The total output of lead, zinc, and silver, largely from Broken Hill, is valued at £3,074,877, a decline of £760,557 from value of production in 1929. Tin output was 590 long tons, valued at £84,800, a decline of £106,399. Gold output was 12,493 oz., a drop of 4,997 oz.

. . . . A tunnel driven at the Mount Bischoff Tin property, at Waratah, Tasmania, has cut a formation 17 ft. wide. Of this, 12 ft. assays 3.4 per cent tin. A drift is now being driven north along the vein. During 1930, extraction and prospecting were confined to leasers. A discovery was made at White Face, where the top of what appeared to be a rich deposit was uncovered. The drainage tunnel, fortunately, was only a short distance away. About 100 ft. of driving was necessary to cut the formation 33 ft. below the point where originally discovered. This discovery, according to J. H. Levings, superintendent, may have an important bearing on the future of the mine, as it is below the present main tunnel and tramway system.

. . . . Broken Hill Proprietary has optioned the Silver Casket lead discovery, near Mount Isa, in northern Queensland.

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Approve Mount Isa Debentures; Mill to Start on May 1

London, March 31.—At the meeting of Mount Isa debenture holders, the company's new issue of £666,667 in Australian-currency debentures, carrying 6 per cent interest, was ratified. This issue will be offered to holders of the first debentures, carrying 8 per cent interest, and shareholders. Leslie Urquhart, chairman of the board of directors of Mining Trust, who presided at the meeting, stated that the first mill unit at the property, in northern Queensland, will start on May 1. A second mill unit and the lead smelter will start operations on May 28. The desilverizing plant in England, which has a capacity of 6,000 long tons of lead monthly, will be ready on July 1. (See Melbourne cable for further information on Mount Isa operations.) Mr. Urquhart stated that a lead shortage was probable, as few producers outside of the United States were working at a profit under present depressed market conditions. This was the reason for the policy of development and equipment on a large scale, which Mount Isa has adopted. It will provide for rapid increases in production in the future, if necessary.

. . . . The report has been circulated here that Randfontein No. 2 North shaft has cut the Main Reef series and that it shows grade and width 50 per cent better than average developments hitherto in the original mine. Randfontein has been hoisting its ore supply through two vertical shafts in the southern half of its property, on the West Rand, Transvaal. The average grade was given in the last ore estimate as 5.8 dwt. over a width of 41 in. Within the last year, the situation at this property has improved. In 1930, the working profit was £436,082, compared with £269,066 in 1929, largely the result of an increase of 1s. 7d. in the grade of ore handled. Hoisting capacity, however, has limited production and development, as was shown in February, when an accident in the south vertical shaft affected output. Even so, the profit for February was £43,593, compared with £25,643 in February, 1930. Sinking of the No. 2 North shaft was started in 1930, after it had been thoroughly repaired, to overcome this deficiency and also to permit more economical development of the northern half of the property. The shaft had been abandoned in 1912 by the old Randfontein Deep company, when it was only down 2,083 ft. It is now down about 4,000 ft. and will be bottomed at 4,600 ft.

. . . . Production has been started at the El Mineral deposit of Panama Corporation, on the north side of Veraguas, Panama. This is an alluvial deposit that will be worked by hydraulic monitors. The gold comes from a gravel deposit, once an extensive river bed, but now covered with lava 100 ft. thick. One large block of 60 acres, averaging about 1,200 ft. wide, has been the center of development. The company is creating an issue of £1,000,000 in debentures, carrying 8 per cent interest. Of this £400,000 will soon be issued. It carries conversion rights and options on additional capital issues.

. . . . Nundydroog Mines, second largest of the gold producers in the Kolar district, Mysore, India, reports a net profit of £100,408 in 1930, compared with £108,081 in 1929. Ore reserves, on the other hand, increased 37,000 tons to a total of 302,827 long tons on Dec. 31, 1930. Developments in the mine were generally satisfactory, the management reports.

. . . . Two of the large German potash producers are going to merge—Kaliwerke Westeregeln and Kaliwerke Siegmundshall. The former company will increase its capital by 1,000,000 marks to a total of 23,200,000 marks to make the amalgamation possible.

■

Outlying Sections of Rand Are Being Examined and Staked

Johannesburg, March 30.—The widespread discussion of a possible gold shortage and the steadily increasing production of the Witwatersrand, which has been markedly accentuated since the beginning of 1930, are focusing world attention on this field, the greatest gold producer of all. Well-known mining engineers, representing important overseas interests, are examining outlying sections of the Witwatersrand formation. Sir Thomas Cullinan and associates have pegged 420 claims in the vicinity of Krugersdorp, in the extreme western

section of the Rand. This area is supposed to contain the lost portion of the Main Reef Leader, which loses its value west of Roodepoort.

. . . . Considerable interest is being taken in the conglomerate gold deposits in the northern section of the Orange Free State. An area about 12 miles long is being prospected. At several places, good assays have been secured. On the Farm Doornkop, a small productive plant is starting work on material from the upper conglomerate beds. According to the official government figures, the only gold production from the Orange Free State in recent years has been 59 oz. in 1928.

. . . . In well-informed quarters, it is believed that the Witwatersrand mines are assured of an adequate supply of native labor for at least eighteen months. Up to the beginning of 1930, production was seriously hampered by the scarcity of workers, but the depression in most commodity prices, which threw many out of work in other sections of South Africa, has resulted in an abundant supply since that date. At the end of February, 209,777 colored workers were employed in the gold mines of the Transvaal, compared with 196,752 at the end of February, 1930.

. . . . Sub Nigel, operating at Heidelberg, south of the Far East Rand, will put additional milling equipment into operation in June. The company's mill is now handling about 31,000 short tons monthly. At the East Geduld property, on the Far East Rand, excellent progress is being made in sinking. The main shaft is now connected with the fourth haulage level. It will probably be bottomed at about 3,000 ft., and is already down more than 2,500 ft.

. . . . Manganese production of Postmasburg, where Manganese Corporation and other companies are at work, is steadily increasing. According to its agreement with the government, this company must ship 200,000 long tons during the first year that the railroad to Koopmansfontein is in operation. The railroad was opened in November, 1930. In December, output was 17,000 long tons, or at an annual rate of 200,000 tons.

■

Roan May Start Within Month

London, April 1.—Only odds and ends remain to be finished at the new Roan Antelope concentrator, in Northern Rhodesia. Operation of the plant (capacity, 5,000 short tons daily) may be started within a month. The mine is ready to supply the mill with its full tonnage. In fact, the speed of opening is increasing the 300,000 tons of ore already on stockpiles at the property.

. . . . Other Rhodesian mines are making excellent progress. Mufulira has recently doubled the rate of development work underground and is also pushing mill construction. Only the first 1,500-ton unit will be completed for the present, because of the unfavorable copper situation. At Chambishi, which may ship ore to the Mufulira mill, shaft-sinking operations have now reached a depth of 242 ft.

Chambishi is now owned by Mufulira Mines, having been exchanged by Rhodesian Selection Trust in return for shares.

Brief Notes

(Continued from page 338)

depth of 1,740 ft. A level will then be established for development purposes. At the Mohawk mine, development of ground near No. 6 shaft yielded favorable results, and prospects are excellent for 1931. No. 1 shaft is nearly exhausted. Although the company produced concentrate containing about 13,300,000 lb. of copper, more than 75 per cent of this was stored instead of being shipped to smelter.

Noranda Mines. High-grade copper ore, with occasional shoots of good-grade gold ore, continues to a depth of 2,700 ft. at the Noranda Mines property, Rouyn, Que., J. Y. Murdoch, president of the company, revealed at the annual meeting in Toronto on March 27. Eighteen holes have been completed in a program of drilling from the 1,475 level, and two additional ones are being put down. They prove ore over a length of 500 ft. on an east-west line in the No. 3 shaft area. One hole, put down at an 80 deg. angle, was in solid sulphides for 1,195 ft., with the exception of two diabase bands. It showed 177 ft. assaying $13.25 in gold and 1.3 per cent copper; 120 ft. assaying $7.78 in gold and 5.9 per cent copper; 100 ft. assaying $12.78 in gold and 13 per cent copper; and 10 ft. of $7.82 in gold and 14.2 per cent copper.

Northern Aerial Minerals Exploration. A party is now at work on the company's claims in the Coppermine River country, 40 miles from the Arctic Ocean. Equipment has been taken in for trenching the frozen overburden, and also for the electrical prospecting of the vein under the direction of representatives of Mason, Slichter & Hay. Trenching will expose the vein at intervals. Where opened up last year, the vein showed 12 ft. of 45 per cent copper ore, and those principally concerned believe that this ore can be mined, transported to the Arctic Ocean, and shipped at a profit.

North Lily Mining. Discovery of gold ore in the fissure zone in the quartzite formation underlying the main orebody was the most important development at the property in 1930, according to J. O. Elton, president of the company, operating near Dividend, Utah. As a consequence, gold production doubled, the grade rising from 0.12 oz. per ton shipped in 1929 to 0.39 oz. per ton in 1930. Long-range exploration has lessened prospects for finding new orebodies within the original property. North Lily is now on a one-shift basis. North Lily-Knight, the development subsidiary with holdings equal to one-third of the total acreage of the Tintic district, confined development to the Big Hill and Dragon properties. North Lily is developing, under lease, the Eureka Bullion, Tintic Bullion, and Baltimore claims.

Park City Consolidated. Ore reserves at the property on Dec. 31, 1930, were estimated at 68,127 short tons by J. J. Beeson, general manager of the company. This includes probable and possible ore. Grade is estimated to be about that mined in 1930—1 per cent lead, 2 per cent zinc, and 20 oz. of silver per ton. The most important new development planned for the immediate future will be extension of a crosscut on the 400 level toward the southern boundary of the property, where three or four veins outcrop on the surface. This work is in the Silver King ore horizon. In addition, development along the Crescent and Roosevelt veins will probably be continued.

Seneca Copper. Results of development work since reopening the mine, in Keweenaw County, Mich., in October, 1928, have exceeded the estimates made by the management at that time. Substantial areas of well-mineralized vein have been made ready for stoping, according to F. R. Kennedy, president of the company. At depth, the work has shown that the downward extensions of orebodies mined in the adjoining Ahmeek and Mohawk properties enter the Seneca property. The lowest levels in both Nos. 1 and 2 shafts disclose the highest-grade ore found in the Seneca property. The management recommends construction of modern sorting equipment at both shafts and erection of a steel headframe at the No. 1 shaft, now timbered. In addition, the company will need increased, compressed-air capacity to the extent of about 7,000 cu.ft. per minute to permit production and development on the scale for which the mine is now ready. At present the property is virtually shut down, pending improvement in the price of copper.

South Tintic Mines. Driving of a tunnel, which will attain a length, eventually, of 5,000 to 6,000 ft., has been started at the property, 3 miles south of the Knight drain tunnel in the Tintic district, Utah. The new tunnel is expected to cut all the important fissures on the property.

Tintic Standard. New orebodies of importance have been disclosed in the limestone areas on the 1,350 level of the mine, near Dividend, Utah, according to E. J. Raddatz, president of the company. Development work in the quartzite formation on the 1,450 level has opened up copper-gold bearing fissures. During 1930, the company drove 7,949 ft. of drifts, 967 ft. of raises, and sunk its shaft 12 ft., a slightly higher footage than in 1929. The churn-drilling program in the eastern section of the company's holdings has now been completed. In addition, Tintic Standard is doing considerable work in the adjoining East Tintic and Eureka Lilly properties, on a lease basis. During 1930, 3,052 ft. of drifts and 233 ft. of raises were completed at these two mines.

U. S. Smelting, Refining & Mining. Exploration and development programs are being actively carried on at the United States and Lark mines, Bingham, Utah, and the Sunnyside mine, Silverton, Colo. Ore reserves have been substantially increased, according to C. A. Hight, president of the company. The grade is better and outlook for the future is good, even though the Sunnyside mine had to be closed because of present low metal prices. Production at both the United States and Lark mines increased in 1930, the Utah state tax commission's report reveals. In 1930, the United States mine extracted 236,700 short tons, containing 17,700 oz. of gold; 1,164,500 oz. of silver; 48,700,000 lb. of lead; 2,850,000 lb. of copper; and 44,000,000 lb. of zinc. The Lark mine produced 79,400 short tons, of about the same grade. In 1929, United States produced 206,000 tons and Lark, 61,200 tons.

Western Empire Mines. The company has completed a mill test of 1,000 short tons of ore from its Royal property, near Hodson, Calif. Flotation is being used experimentally.

MARKET AND FINANCIAL NEWS

Phelps Dodge 1930 Earnings 6 Per Cent of 1929 Mark

Net income of Phelps Dodge and subsidiaries in 1930 was $515,173, after depreciation and tax charges, compared with $9,601,212 in 1929—or about a 94 per cent decline. As a result of the acquisition of Nichols Copper and National Electric Products, the actual volume of business more than doubled, proceeds from sales having been $83,969,979, compared with $38,705,671. Operating profit in 1930 was $3,035,391. This compares with $12,296,769 in the preceding year.

Production from Phelps Dodge mines in 1930 was 141,662,995 lb. of copper; 151,413 lb. of lead; 12,620 lb. of manganese; 768,334 oz. of silver; and 20,839 oz. of gold. Custom ores produced 2,190,257 lb. of copper; 4,971,874 lb. of lead; 799,056 oz. of silver; and 3,200 oz. of gold. Sales of copper from the company's producing properties amounted to 220,253,379 lb., exhausting the copper on hand at the end of 1929. Average price received by the company for copper sold in 1930 was 11.309c. per pound, f.o.b. New York. Laurel Hill and El Paso refineries of Nichols Copper produced 425,000,000 lb. of refined copper from Phelps Dodge and custom blister; 4,150,000 oz. of silver; and 140,000 oz. of gold.

On Dec. 31, 1930, the balance sheet showed total assets of $285,424,369, compared with $242,926,038 at the end of 1929. Current assets were $45,032,270 and current liabilities were $15,287,220. The surplus was $62,580,621, which compares, after readjustment for absorption of new companies, with $67,841,464. A special reserve of $3,000,000 has been created for contigencies and for possible expenditures in co-ordinating and developing newly acquired units of the company.

■

Patiño M. & E. Loss Large

Net loss of Patiño Mines & Enterprises, after all charges, was $767,159 in 1930, compared with a net profit of $4,573,239 in 1929. Operating profit in 1930 was $850,268 on a production valued at $7,338,228. Production was 17,358 metric tons of fine tin. At the end of 1930, Patiño held 5,896 long tons of tin, valued at £116 7s. 2d. a ton. Total assets on Dec. 31, 1930, were $37,673,368, of which $7,956,662 was inventories, $760,991 was cash, and $3,266,133 was investments. Liabilities included $1,891,165 of bank drafts and $941,697 in accounts payable.

E.&M.J. Weighted Index of Non-Ferrous Metal Prices

100 Is Composite for 1922-3-4

1920	130.92
1921	80.80
1922	93.94
1923	103.43
1924	102.63
1925	111.74
1926	107.74
1927	97.54
1928	99.78
1929	110.33
1930	82.87

	1929	1930	1931
January	106.86	102.94	67.62
February	110.37	102.28	65.90
March	122.71	99.93	68.03
April	116.36	93.24	.....
May	110.03	83.41	.....
June	109.66	78.57	.....
July	109.58	74.58	.....
August	109.81	74.41	.....
September	109.29	73.42	.....
October	108.52	69.63	.....
November	106.09	71.39	.....
December	104.75	70.65	.....

■

Calumet & Hecla Consolidated Showed Net Loss in 1930

A net profit, after all charges, of $5,326,932 in 1929 was turned into a net loss, after all charges, of $6,749, by Calumet & Hecla Consolidated Copper, leading copper producer in Michigan, as a result of the decline in the price of the metal and its own curtailed production. Before depletion or depreciation deductions, the company did show an operating profit of $2,058,755, out of gross receipts of $8,890,043. Of this, $8,609,139 was from sales of copper. Production in 1930 was 105,740,000 lb., against 123,850,000 lb. in 1929.

Copper on hand at Dec. 31, 1930, showed a large increase over stocks at the end of 1929, being $6,826,690, compared with $2,982,165. Total assets at the end of 1930 were $52,642,281; current assets were $14,729,432; current liabilities were $1,579,529; investments in other companies were $1,767,111; and the company's capital stock and surplus account stood at $50,602,752.

Alaska Juneau Net Drops With Lower Production

Net profit, before depletion, earned by Alaska Juneau Gold in 1930 was $831,086, compared with $1,025,208 in 1929. The decrease resulted largely from increased expense in connection with outside exploration work and the lower grade of ore handled. Although actual tonnage mined was greater in 1930 than in 1929, the value of the metals produced dropped from $3,627,247 in 1929 to $3,551,950 in 1930. Operating profit in 1930 was $1,262,594, before bond interest or other charges.

All bonded and other indebtedness of every kind has been paid off by the company. Dividend payments at the rate of 10c. quarterly on the company's 1,443,000 shares outstanding have been started. As present earnings are more than ample for this purpose, and for carrying out plans for securing additional mining properties, the board of directors has authorized that any surplus earnings be invested in the shares of the company, F. W. Bradley states in the annual report. The company is now exploring two groups of claims in the Juneau district on a 50-50 basis with Treadwell Yukon.

The balance sheet on Dec. 31, 1930, showed a surplus of $1,275,395. Total assets were put at $17,591,242, of which $703,152 was in current assets. Current liabilities totaled $154,000. A depreciation reserve of $1,670,207 has been built up. On Dec. 31, 1929, total assets were $17,485,150; current assets, $477,570; current liabilities, $151,097; depreciation reserve, $1,513,074; and surplus, $436,289. At that time the company had a funded debt of $893,400, all of which was paid off in 1930.

■

Chief Consolidated Mining

Net loss, after payment of taxes and bond interest, was $69,901 for Chief Consolidated in 1930, compared with a profit of $150,659 in 1929. The company showed an operating loss of $9,406 on a gross production of $782,210. Output of lead was 3,500 tons; silver, 998,605 oz.; and gold, 16,702 oz. At the end of the year the company had a net surplus of $3,349,441, of total assets of $5,781,971. Current assets were $104,032; current liabilities $178,937.

U. S. Smelting Net in 1930 Compares Well With 1929

One of the best showings among base-metal producers in 1930 was made by U. S. Smelting, Refining & Mining. In 1930 it reported a net profit of $3,699,-656, after depreciation, depletion, and taxes, compared with $4,818,867 in 1929—or less than a 25 per cent decline. Operating profit in 1930 was $6,599,802; in 1929, $7,408,863. Production figures are not given.

The earned-surplus account on Dec. 31, 1930, shows no change from the previous year, surplus earnings after payment of dividends having been diverted to additional reserves for amortization of property. The surplus is $17,629,243. Total assets are shown in the balance sheet as $79,473,259, of which $14,154,095 is current assets. Current liabilities are $2,507,096.

During 1930, the company purchased 56,700 shares of its common stock in the open market at an average cost of $21.68. This stock is being held in the treasury for corporate purposes. Outstanding common stock now totals 563,-855 shares.

■

Aluminum Company Earnings

In 1930, the Aluminum Company of American made a net profit, after all charges, of $11,672,726, compared with $25,318,050 in 1929. Operating profit in 1930 was $18,784,344. Tonnage sold in 1930 was about two-thirds of that sold in 1929, A. V. Davis, chairman of the board of directors of the company, states. The price of aluminum in 1930 was 1c. a pound less than in 1929. Total assets of the company on Dec. 31, 1930, were $242,986,667, of which $38,282,486 is inventories; $3,472,496, cash; $33,-194,866, accounts, notes receivable, and market securities; and $51,984,922 investment in subsidiary and affiliated companies. The surplus on Dec. 31, 1930, was $27,971,219, compared with $25,938,283 at the end of 1929.

■

Miami Copper Profit Down

Net profit, after depreciation and tax charges but before depletion, earned by Miami Copper in 1930 was $726,713, compared with $4,151,081 in 1929. Gross operating revenue was $8,425,632 and operating profit was $1,123,687 in 1930. The average sales price received, f.o.b. New York, was 14.87c. a pound, considerably better than the average price of copper during the year. Production was 67,124,596 lb. of copper at a cost of 10.878c. a pound, including repairs, renewals, and upkeep of plant but not the direct charge for depreciation.

Miami's balance sheet shows total assets of $26,219,023, on Dec. 31, 1930, of which $2,589,027 represents cash, investments, and accounts receivable. Copper in process and on hand at the close of the year was valued at $1,976,-639, compared with $3,772,256 on hand at the close of 1929. Current liabilities of Dec. 31, 1930, were $1,008,722 and the earned surplus was $8,709,056, compared with $10,844,490 at the close of 1929.

■

Inspiration Consolidated

After deduction of depreciation, taxes, and interest charges, Inspiration Consolidated Copper earned $273,759 in 1930, compared with $4,753,802 in 1929. Sales of copper totaled $9,769,174 in 1930, on which an operating profit of $998,136 was made. Comparable figures in 1929 were $13,182,122 and $5,923,982. Copper production in 1930 was 65,-606.664 lb.

At the close of 1930, the Inspiration balance sheet showed total assets of $45,526,484. Of this, $4,391,668 was current assets, including $2,154,676 for finished copper on hand—at cost (9.6671c. a pound after depreciation charges in 1930). At the close of 1929, Inspiration had copper inventories of $4,850,744. Current liabilities on Dec. 31, 1930, were $1,102,908 and the earned surplus was $8,160,933, a reduction from the earned surplus of $9,660,124 at the end of 1929. At the close of 1930, the net amount of 6½ per cent gold notes outstanding was $3,579,500.

■

CONIAGAS MINES, now largely a holding company but once a large silver producer, reports net profits of $6,434 in 1930. This increases the company's surplus to $2.004,434. Current assets total $1,733,308 of total assets of $2,058,264. Current liabilities are $15,121.

Mining Company Dividends for March, 1931

Companies in the United States	Situation	Per Share	Total Disbursements
American Metal, 6% pfd.	Various	$1.50 Q	103,014
American Smelting & Refining, 6% pfd.	Various	1.50 Q	262,500
American Smelting & Refining, 7% pfd.	Various	1.75 Q	875,000
Bunker Hill & Sullivan, s.l.	Idaho	0.25 M	122,625
Eureka Standard, g.s.	Utah	0.03 Q	45,000
Federal Mining & Smelting, pfd.	Idaho, Oklahoma	1.75 Q	126,000
Golden Cycle, g.	Colorado	0.40 Q	66,000
Granite Gold Mining	Colorado	0.01 Q	16,500
Hecla Mining, l.s.	Idaho	0.10 Q	100,000
Homestake Mining, g.	S. Dakota	0.50 M	125,800
(a) Iron Cap Copper, pfd.	Arizona	0.15 Q	24,315
Mayflower Associates, h.	Various	0.50 Q	136,500
National Lead	Various	1.25 Q	387,289
National Lead, pfd. "A"	Various	1.75 Q	426,433
Nevada Consolidated Copper	Nev., Ariz., N. M.	0.25 Q	1,214,312
St. Joseph Lead	Various	0.50 Q	975,764
Tennessee Corporation, c.	Tennessee	0.12½Q	198,656
Texas Gulf Sulphur	Texas	1.00 Q	2,540,000
Tintic Standard Mining, s.l.	Utah	0.10 Q	115,332
Utah Copper	Utah	2.00 Q	3,245,990
Companies in Other Countries			
Chile Copper	Chile	0.50 Q	2,207,751
Hollinger Consolidated Gold	Ontario	0.05 Q	246,000
International Nickel, n. c.	Ontario	0.15 Q	1,768,297
Lake Shore, g.	Ontario	0.30 Q	300,000
McIntyre-Porcupine, g.	Ontario	0.25 Q	199,500
Total			$15,828,578

s, silver; l, lead; c, copper; n, nickel; g, gold; q, quicksilver, h, holding company; Q, quarterly; M, monthly; X, extra; (a) Paid in February, omitted from our list.

Financial Notes

CLIFF MINING, which owns a property in Keweenaw County, Mich., reports a balance of current assets of $123,394 at the end of 1930. During the year the company spent $169,545 in exploration work. Gross receipts were $5,350, largely interest.

LA SALLE COPPER, which owns a copper property in Michigan, reports a profit of $4,959 during 1930, which increased the company's current assets to $342,618 and the excess over current liabilities to $339,916. No copper was produced during the year, the income being derived from interest and rentals.

PARK CITY CONSOLIDATED, operating a developing silver mine at Park City, Utah, reports an excess of disbursements over receipts, in 1930, of $37,026. The company produced ore from which a net return of $66,233 was received. The balance sheet shows total assets of $1,249,816, current assets of $18,508, and current liabilities of $14,782.

ST. JOSEPH LEAD stockholders have been asked to ratify an issue of $10,000,-000 of convertible debentures. The debentures will be offered to them at the rate of $100 for every twenty shares of stock held. The purpose of the financing will be to reimburse the company in part for expenditures of $21,607.427 in mine development during the last four years.

CALIFORNIA-AHUMADA MINING reports a net profit of $66,372 in 1930, after depreciation and Mexican tax payments. The year was the first of regular operation of the company's lead properties in Mexico. Net value of production was $216,441, and the operating profit was $78,511. At the end of the year the company had total assets of $2,009,441, of which $54,929 was in current assets.

1930 Nickel Consumption Totaled 88,000,000 Lb.

World's consumption of nickel in all forms during 1930 reached a total of 88,000,000 lb., against 136,000,000 lb. in 1929, and 117,000,000 lb. in 1928, according to Robert C. Stanley, president of the International Nickel Company of Canada, Ltd. The world-wide industrial depression, he told stockholders at the annual meeting of the company, was responsible for the substantial decline in the demand for nickel. As for 1931, Mr. Stanley believes that the outlook is favorable and expects that the trend in nickel sales will be reverse of that recorded in 1930.

Sales of nickel from the Port Colborne, Canada, and Clydach, Wales, refiners of International Nickel in 1930 amounted to 56,934,612 lb., compared with 95,394,808 lb. in 1929, a decrease of 40 per cent. Sales of products produced at the rolling mills of the company located at Birmingham, England, and Huntington, W. Va., and the foundry in Bayonne, N. J., totaled 26,454,916 lb., against 41,094,851 lb. in 1929, a decrease of 36 per cent. The sales of nickel in all forms, including alloys, amounted to 75,284,352 lb. in 1930.

■

Seneca Copper Mining

In 1930, Seneca Copper recorded a net loss, after all charges including bond interest, of $721,782, compared with a net loss in 1929 of $287,970. The operating loss in 1930 was $460,274. Production sold for a total of $363,440 and the company's inventory at the end of the year showed $120,072 in copper on hand. The output in 1930 was 4,857,795 lb., compared with 2,999,882 lb. in 1929. At the close of the year, the company reported total assets of $6,448,983, of which $138,021 was current assets. Bonded indebtedness totaled $1,482,500 and operating and development deficit $1,475,634.

■

Iron Cap Records Loss

After charges for depreciation and taxes, Iron Cap Copper, now chiefly a holding company with interests in the Christmas mine, in Arizona, recorded a net loss of $66,612 in 1930, compared with a net profit of $31,988 in 1929. The company sold metals valued at $663,489, which is entirely from the Christmas mine, and made an operating loss of $14,403. Christmas copper production was 6,925,735 lb. The Iron Cap balance sheet shows total assets of $2,969,018, of which $523,948 is current assets. Surplus on Dec. 31, 1930, was put at $218,263. Current liabilities on that date were $335,011 and outstanding bond indebtedness was $23,460.

Financial Notes

CALLAHAN ZINC-LEAD reports net loss, before depletion or depreciation, of $65,910 in 1930. In 1929, net loss was $57,770. Net value of 1930 production was $50,636 and cost of development at the Galena mine was $99,474. Total current assets on Dec. 31, 1930, were $264,610 and liabilities were $6,738.

HUDSON BAY MINING & SMELTING reports a net profit, before depreciation or depletion, of $2,916 on production up to Dec. 31, 1930. This profit was registered after deduction of metal inventories from net operating costs. Sales of metals were $497,163. The company's total assets are put at $33,447,969, of which $1,628,663 is current assets. Current liabilities are $945.053 and funded indebtedness $5,000,000.

■

Utah Copper Valuation Less

The tax law of Utah, which values mines at three times their net profits in the preceding year, is responsible for an apparent shrinkage from $93,242,604 in 1929 to $23,967,617 in 1930 in the tax valuation of Utah Copper, the state's largest individual tax payer. In 1929, net profits were reported at $26,256,755, and the mine valuation for the year was correspondingly $79,580,265. Plants and surface rights were put at $13,662,339. The drop in the price of copper and in production, which was more than 50 per cent in 1930, reduced the net profit to $3,587,558 and the valuation of the mine to $10,762,673—although actually the company's reserves are sufficient for more than thirty years of capacity operation. Plant valuation in 1930 was $13,204,944. As a result of this drop in valuation, the company will pay $1,450,000 less in taxes in 1931.

■

Benguet Consolidated Profit

Net profit of Benguet Consolidated, operating a gold mine in the Philippine Islands, was 2,178,959 Philippine pesos in 1930, compared with 1,664,720 pesos in 1929. The increase resulted largely from a larger income from the company's subsidiary, Balatoc Mining, which increased its dividend payments in 1930. Actual operating profit at Benguet in 1930, before depreciation or depletion, was 1,814,174 pesos, compared with 1,749,144 in 1929. Gross value of bullion production was 3,546,952 pesos ($1,773,476). The balance sheet as of Dec. 31, 1930, showed total assets of 4,286,584 pesos, of which 1,668,799 pesos was current assets. The company's investments are valued at 1,412,948. Dividend payments in 1930 were 2,400,000 pesos, reducing the surplus at the end of the year to 431,408 pesos, compared with 706,923 at the beginning. Current liabilities were 155,176 pesos.

Tintic Standard Mining Net Lower in 1930

The increased production and earnings from Eureka Standard, its gold-mining subsidiary, enabled Tintic Standard to make a comparatively good showing in 1930. Consolidated net profit, after depreciation and taxes, totaled $690,439, compared with $1,382,847 in 1929. If the minority interest in Eureka Standard is added to the 1930 showing, however, the net profit carried to surplus totals $796,055. Total value of production was $3,584,092, compared with $4,265,337. Output included 34,394 oz. of gold, 2,806,988 oz. of silver, 31,521,046 lb. of lead, and 852,400 lb. of copper. Of the gold, 30,540 oz. was contained in Eureka Standard ore, which also carried some of the other metals.

The 1930 balance sheet shows total assets of $11,032,936. This includes $2,856,129 of current assets and $1,224,413 of investments (largely in the company's subsidiaries). Current liabilities total $126,562. The company's surplus on Dec. 31, 1930, was $8,320,099, compared with $8,623,246 at the end of 1929. Net profit of Eureka Standard was $297,091 on a production of $793,412.

■

American Zinc Earnings

In 1930, American Zinc, Lead & Smelting recorded a net profit of $190,623, after depreciation, depletion, interest charges, and taxes. Comparable profits in 1929 was $587,618. Operating profit on sales, before addition of other income or deduction of administrative, selling, and other expenses, was $849,875 in 1930; $1,323,317 in 1929. Net sales were $6,638,254. On Dec. 31, 1930, the balance sheet showed an earned surplus of $1,093,648, compared with $1,264,504 at the close of 1929. Total assets were put at $8,257,053, of which $2,521,417 was current assets. Total inventories, including metal, ores, and materials, were $1,778,303. Current liabilities at the close of 1930 were $816,959.

■

Isle Royale Copper

In 1930, Isle Royale Copper, small Michigan producer, recorded a net loss of $343,667, compared with a net profit of $197,469 in 1929. Operating loss in 1930 was $148,337 on copper sales of $1,105,015. Production was 10,659,413 lb. of copper. Stocks of copper on hand increased from $343,797 at the beginning of the year to $454,176, at the end. The company's balance sheet showed total assets of $6,462,014, of which $1,300,248 was current assets. Current liabilities were $133,814 on Dec. 31, 1930.

Nevada Con. 1930 Income 10 Per Cent of 1929

Net income of Nevada Consolidated, after all charges except depletion, was $1,860,733 in 1930, $17,493,977 in 1929. The annual report indicates a gross operating revenue of $17,978,778, on the basis of total 1930 production, and an operating profit of $2,397,041. Production in 1930 was 141,980,173 lb. of copper, 29,165 oz. of gold, and 123,746 oz. of silver. Cost of producing copper, after all charges except depletion and federal taxes, was 10.85c. a pound, compared with 9.53c. in 1929. In the last quarter, however, costs were reduced somewhat and the total, before depreciation, depletion, or taxes, was 8.96c. a pound.

The Nevada balance sheet showed total assets of $83,220,912 on Dec. 31, 1930, of which $24,010,188 was current assets. There was a reduction in the value of metals on hand and in transit from $15,711,215 at the end of 1929 to $9,894,534. This is partly the result of the fact that at the end of 1929, unsold copper was carried at 13.5c. a pound and at the end of the year it was carried at 10.5c. Current liabilities on Dec. 31, 1930, were $4,465,393 and the earned surplus was $12,631,961. On Dec. 31, 1929, earned surplus was $22,792,428.

■

First Year of Falconbridge

The report of Falconbridge Nickel for 1930 covers the first year of actual production by the company. Because of the large amount of its products tied up in stocks and in process at its refinery, a demonstration of the company's earning power was not possible. Sales of copper totaled only $20,803 and of nickel, $58,642. If metal in stock and in process, including precious metals is included, the company's 1930 production would have been worth $991,962 at the prices actually received for the output. As costs totaled $783,805, a profit of $208,257 before depreciation or depletion is indicated. On Dec. 31, 1930, the company's assets totaled $6,515,280 and liabilities, $6,778,453, indicating a deficit of $263,173. Current liabilities were $83,072 and current assets, not including metal stocks, $228,681.

■

North Lily Mining Income

Net profit of North Lily Mining, after providing for depreciation and federal taxes, was $160,783 in 1930, compared with $792,914 in 1929. Production was sold for $832,248, compared with $1,647,-108 in 1928. Operating profit in 1930 was $193,843. Production was 15,843,-836 lb. of lead, 357,673 lb. of zinc, 439,443 oz. of silver, and 21,944 oz.. of gold from 55,717 short tons of ore. Only gold output showed an increase over the record for 1929. On Dec. 31, 1930, the company's balance sheet showed total assets of $1,241,667, including $503,190 in current assets, $521,101 in fixed assets, and $217,377 in other assets. Current liabilities were Dec. 31, 1929.
$32,919 and the surplus was $1,029,645, compared with a surplus of $989,859 on Dec. 31, 1929.

■

Ohio Copper Loss Greater

Although the cost of copper production decreased from 14.06c. to 13.26c. a pound, Ohio Copper showed a greater loss in 1930 than in 1929 as a result of the drop in the price of the metal. The net loss, after all charges, including bond interest, was $117,905, compared with $41,030 in 1929. Operating loss in 1930 was $1,856 on a production of 2,048,369 lb., compared with an operating profit of $88,099 in 1929. The company ha. total assets of $4,283,128, and a deficit of $341,052 as liabilities total $4,630,179. Funded debt outstanding is $179,300.

■

Electrolytic Copper Output

Production of electrolytic copper in 1930 amounted to 1,681,315 tons, against 1,969,432 tons in 1929, according to the American Bureau of Metal Statistics. Production of copper by the electrolytic process accounts for about 87.5 per cent of the total output of the metal. Figures showing the world's total on a refined basis for 1930 are not yet available.

Electrolytic copper production, in short tons, for 1929 and 1930 is estimated as follows:

	1929	1930
United States	1,444,392	1,155,459
Rest of world	525,040	525,856
Totals	1,969,432	1,681,315

The total for 1930 is subject to revision, one comparatively small producer's output being estimated.

Investigate Russian Asbestos

An investigation of alleged unfair practice in the importation and sale of Russian asbestos in the United States has been ordered by the U. S. Tariff Commission. A hearing will be held in Washington May 19.

The action resulted from complaints by the Bear Canyon Asbestos Company, Ambler, Pa., and the Regal Asbestos Mines, Inc., New York, owners and operators of mines in Arizona. The Amtorg Trading Corporation, New York, and Asbestos Limited Inc., of Millington, N. J., were named in the complaints.

■

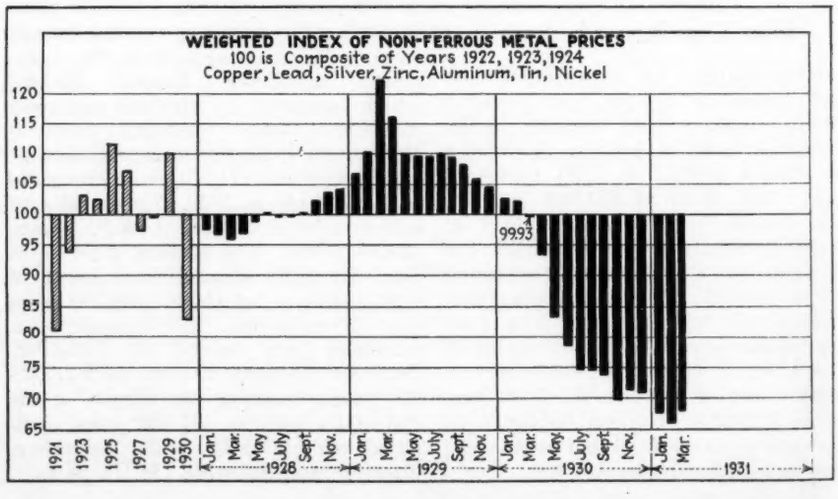
Engels Copper Shows Loss

In 1930, Engels Copper realized a net loss of $236,630, after all charges, in its mining and milling operations, compared with a net profit of $115,626 in 1929. The mine was shut down in the middle of 1930. An operating profit of $11,441 was earned in the first six months of the year, compared with an operating profit of $663,527 in 1929. Gross value of metal production in 1930 was $643,291. It included 4,072,628 lb. of copper. The company had total assets of $7,939,199 on Dec. 31, 1930, of which $125,527 is current assets. Current liabilities totaled $6,776 and the surplus on Dec. 31, 1930, was $990,142.

■

Balatoc Mining Earnings

Net profit of Balatoc Mining, operating a property in the Philippine Islands, 1,046,550 Philippine pesos in 1930, compared with 1,247,115 pesos in 1929. The company handled a lower grade ore, the average being 47.95 pesos per ton, compared with 56.72 per ton in 1929, thus accounting for the lower earnings. Total bullion production was valued at 2,251,793 pesos.



Slow Demand for Major Metals Results in Renewed Weakness in Prices

New York, April 9, 1931.—The movement of prices for all of the major metals except lead was downward in the last two weeks. Disappointment over the slow rate of improvement in general business activity caused fabricators to curtail purchasing operations severely. The continued weakness in security prices also acted as a damper. In order to move even a moderate tonnage of copper and zinc, sellers had to make further price concessions, and, in the case of the latter, extend deliveries well into the future. Trade reports have been more encouraging, but operators in the non-ferrous metal field are slowly giving up the idea that the revival in business in the first half of the year will be more than seasonal.

The *E.&M.J.* weighted index number of non-ferrous metal prices for March was 68.03, against 65.90 in the preceding month. The gain for March reflects the improvement in business and consequent strength in prices of copper, tin, and silver, which was so evident in the first half of that month.

Copper Unsettled

The limited business booked in the last two weeks in the domestic market was confined almost wholly to sales by custom smelters who felt that they could not afford to accumulate supplies under existing conditions. This selling resulted in business at prices ranging from 9½c. to 9¾c. per pound, delivered Connecticut, the inside figure equaling the low attained on two previous occasions since the advent of the present depression. The large copper-producing groups maintained their nominal quotation of 10c. throughout the period. The export quotation also held on this basis —namely 10.30c., c.i.f. European ports —but buying for foreign account was quiet.

Custom smelters experienced no difficulty in selling copper at 9½c., and it was quite evident that a substantial tonnage could have been sold at that figure. Just as soon as custom smelters sold the modest tonnage they had to offer, the price steadied, and on the last day of the period under review business went through at 9¾c., Connecticut basis.

Further attempts have been made to arrive at some solution of the problem of the custom smelters selling their intake in a dull market without wrecking the copper price structure, but no plans have yet been found acceptable. The fact that the production of some fairly large interests, like International Nickel, Cerro de Pasco, Granby, and Consolidated Coppermines, is or may be sold through the custom smelters' agencies, complicates the situation.

Unless general industrial activity throughout the world shows improvement soon, it is felt in copper circles that steps will have to be taken to again reduce output.

A hearing was held in Washington on March 31, before the Tariff Commission, in connection with the study in copper costs. The result of the study is expected to be ready for the next session of Congress.

Lead Steady

Demand for lead was quiet, but showed some improvement toward the close of the fortnight. There was a fair demand for April metal, indicating that consumers have been restricting purchases to a minimum because of the general unsettlement in the lead situation, especially abroad.

The Lead Producers' Association met in London to consider a plan for curtailing foreign production from 15 to 20 per cent. Numerous objections to the proposals were raised, and, at times, the success of the plan seemed doubtful. This accounted for the irregularity in London prices. Cables received here April 8 indicate that the difficulties have virtually been settled and official announcement of the agreement to restrict output is expected shortly.

The New York market held at 4.50c., with St. Louis showing little variation from the 4.25c. basis throughout the period.

Zinc Offerings Increase

Zinc was offered quite freely and the market developed further weakness. Heavy business was booked in the calendar week ended March 28, the sales, mostly Prime Western, amounting to about 7,000 tons. Few sellers shared in this business, and most of the buying was for account of large sheet mills and galvanizers. Demand since then has been quiet, and buyers experienced no difficulty in obtaining zinc at 3.90@3.95c. St. Louis basis for delivery over the second quarter of the year.

Tin Below 26c.

On rumors that the tin plan is meeting with some difficulties, speculative holders of the metal in London became nervous and liquidated part of their commitments. This weakened prices and uncovered quite a few stop-loss orders. Compared with prices that prevailed two weeks ago, the New York market suffered a decline of almost 2c. per pound. Buying here was quiet. The world's visible supply of tin on the last day of March was 48,607 long tons, against 49,339 long tons a month previous.

Silver Declines

Uncertainty over the general business situation in the Far East restricted buying of silver appreciably, and the market again turned weak, the price in New York falling about 3c. per ounce in the last two weeks. Selling by China was in evidence throughout the period.

Following a conference lasting almost an hour on April 8 between Henry L. Stimson, Secretary of State, and Montagu Norman, governor of the Bank of England, administration spokesmen let it be known definitely that the United States would make no move to call an international conference on silver. The U. S. Senate Committee on Foreign Affairs recently voted in favor of calling such a conference.

Average Metal Prices for March, 1931

COPPER:	
Electrolytic, refinery	9.854
London Standard Spot	44.818
London Electrolytic, Bid	47.699
LEAD:	
New York	4.527
St. Louis	4.276
London Spot	13.128
London Forward (*a*)	13.349
SILVER:	
New York	29.192
London	13.524
Sterling Exchange	485.596
ZINC:	
St. Louis	4.002
London Spot	12.190
London Forward	12.676
TIN:	
Straits	27.065
London Standard Spot	121.852
QUICKSILVER	100.423
ANTIMONY	7.127
PLATINUM, Refined	29.538
CADMIUM	55.000
ALUMINUM, 99 Per Cent Plus	23.300

(a) Subject to confirmation.

Other Metals

ALUMINUM—Per lb., delivered, Alcoa commercial and mill ingot, 99 and 98 per cent, 23.30c.; Alcoa No. 12 alloy, 22c.; metallurgical ingot, 94 per cent plus, 23.30c.; 98@99 per cent, 22.90c. London, £85, less 2 per cent, per long ton, for 98 per cent ingots and bars.

ANTIMONY — Per lb., duty paid, Chinese spot, 7c., futures, 6.90c. Cookson's "C" grade, spot, 12c. "C.M.C." grade, 99.9 per cent, spot, 11c.

BISMUTH—Per lb., $1.50.

CADMIUM—Per lb., 55c.

IRIDIUM—Per oz., $125@$135 for 98 @99 per cent sponge and powder.

NICKEL — Per lb. electrolytic cathodes, 35c.; shot and ingot made from remelted electro, 36c., for single lots of spot metal. London, per long ton, £170 @£175, as to quantity.

PALLADIUM—Per oz., $19@$21.

Daily Prices of Metals

March April	Electrolytic Copper	Straits Tin	Lead		Zinc
	Refinery	New York	New York	St. Louis	St. Louis
19	9.775	27.40	4.50	4.25	4.00
20	9.775	27.30	4.50	4.25	4.00
21	9.775	27.25	4.50	4.25	4.00
23	9.775	27.15	4.50	4.25	4.00
24	9.775	26.95	4.50	4.25	3.975
25	9.775	27.05	4.50	4.25	3.95@4.00
Av'ge	9.775	27.183	4.500	4.250	3.992
26	9.775	27.00	4.50	4.25	3.95
27	9.650	26.80	4.50	4.20@4.25	3.95
28	9.525	26.85	4.50	4.25	3.95
30	9.525	26.50	4.50	4.25	3.95
31	9.525	26.75	4.50	4.25	3.95
1	9.525	26.625	4.50	4.20@4.25	3.95
Av'ge	9.588	26.754	4.500	4.242	3.950
2	9.425	26.50	4.50	4.25	3.95
3	9.275	26.50	4.50	4.25	3.95
4	9.275	26.50	4.50	4.25	3.95
6	9.275	26.50	4.50	4.25	3.95
7	9.275	25.85	4.50	4.25	3.95
8	9.525	25.45	4.50	4.25	3.90
Av'ge	9.342	26.217	4.500	4.250	3.942

Average prices for calendar week ended March 21, 1931, are Copper, 9.775c.; Straits tin, 27.321c.; New York lead, 4.500c.; St. Louis lead, 4.250c.; Zinc, 4.000c.; Silver, 30.125c.

Average prices for calendar week ended March 28, 1931, are Copper, 9.713c.; Straits tin, 26.967c.; New York lead, 4.[illegible]; St. Louis lead, 4.246c.; Zinc, 3.967c.; Silver, 29.354c.

Average prices for calendar week ended April 4, 1931, are Copper, 9.425c.; Straits tin, 26.563c.; New York lead, 4.[illegible]00c.; St. Louis lead, 4.246c.; Zinc, 3.950c.; Silver, 28.063c.

Silver, Gold and Sterling Exchange

March	Sterling Exchange "Checks"	Silver		Gold London	Mar. Apr.	Sterling Exchange "Checks"	Silver		Gold London
		New York	London				New York	London	
19	4.85 11/16	29 3/4	13 5/16	84s 10½d	30	4.85 11/16	28 3/4	13 3/16	84s 10¾d
20	4.85 5/8	29 5/8	13 3/16	84s 10¾d	31	4.85 5/8	28 3/4	13 3/16	84s 9¾d
21	4.85 3/4	29 1/4	13 9/16	.....	1	4.85 5/8	28 1/8	12 15/16	84s 10¼d
23	4.85 3/4	29 1/4	13 9/16	84s 11d	2	4.85 11/16	27 5/8	12 7/8	84s 10¼d
24	4.85 13/16	30	13 13/16	84s 10½d	3	4.85 5/8	27 5/8	Holiday	Holiday
25	4.85 3/4	29 1/4	13 1/2	84s 10½d	4	4.85 3/4	27 1/2	Holiday	.....
26	4.85 3/4	29 3/8	13 1/2	84s 10½d	6	4.85 3/4	27 1/2	Holiday	Holiday
27	4.85 9/16	29	13 3/8	84s 10¾d	7	4.85 3/4	27 1/4	12 11/16	84s 10¼d
28	4.85 9/16	29 1/4	13 1/2	.....	8	4.85 5/8	27 3/8	12 5/8	84s 10½d

Average for week ended March 25: Silver, 29.521c., Sterling Exchange, $4.85729.
Average for week ended April 1: Silver, 28.875c.; Sterling Exchange, $4.85635.
Average for week ended April 8: Silver, 27.500c.; Sterling Exchange, $4.85698.

New York quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, basis of 925 fine. Sterling quotations represent the demand market in the forenoon. Cables commanded three-sixteenths cent premium on Jan. 7, 1931.

London

March April	Copper Standard		Electrolytic	Tin		Lead		Zinc	
	Spot	3 M		Spot	3 M	Spot	3 M	Spot	3 M
19	44½	45	47½	122¾	124¼	13¼	13⅜	12¼	12 11/16
20	44 13/16	45 7/16	47½	123	124½	13	13 3/16	12 5/16	12⅜
23	44⅝	45	47½	122¾	124¼	12 11/16	13 1/16	11 15/16	12⅜
24	44¼	44 11/16	47	121½	122¾	11 11/16	12	11½	11⅞
25	43 13/16	44½	46½	121½	123	12⅜	12⅞	11 13/16	12¼
26	43½	44⅛	46	121	122½	12	12⅛	11⅝	12½
27	43 1/16	43¾	45¾	120⅜	121⅞	12⅛	12 5/16	11⅜	11⅞
30	43 1/16	43¾	45¾	119½	121	12⅜	12 5/16	11½	12
31	42 15/16	43⅝	45½	119¼	120¾	12 1/16	12 5/16	11⅝	12 1/16
1	43⅜	44 1/16	45½	119¾	121¼	12⅛	12⅜	11¾	12 5/16
2	43⅛	43¾	45½	119¼	120⅝	12¾	12 15/16	12	12 9/16
3				Holiday					
6				Holiday					
7	42 13/16	43½	45½	117¼	118⅝	12 7/16	12¾	11 13/16	12⅜
8	42 9/16	43¼	44½	113⅞	115⅜	12¼	12 11/16	11 11/16	12 5/16

Prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

PLATINUM—Per oz., official price of leading interest was reduced from $28 to $26 on April 8. Cash transactions between dealers several dollars less.

QUICKSILVER—Per 76-lb. flask, $102. Business fair and prices firmer. London reports the market unchanged at £22 7s. 6d. per flask.

Chromium, Cobalt, Magnesium, Manganese, Molybdenum, Osmium, Radium, Rhodium, Ruthenium, Selenium, Silicon, Tantalum, Tellurium, Thallium, and Tungsten are unchanged from prices given in the March 23 issue.

Metallic Ores

Prices in tons of 2,000 lb., or in "Units" of 20 lb., unless otherwise stated.

CHROME ORE—Per long ton, c.i.f. Atlantic ports, Russian and Indian ores, 46 to 48 per cent CR_2O_3, $19.

MANGANESE ORE—Per long ton unit of Mn, c.i.f. North Atlantic ports, cargo lots, exclusive of duty of 1c. per lb. of Mn contained, Indian, 48 to 50 per cent, 26@27c.; South African, 52 to 54 per cent, 25@26c. Caucasian, 52@55 per cent, 25@26c. per long ton unit.

TUNGSTEN ORE—Per unit of WO_3, New York: Chinese wolframite, duty paid, $11.25. Bolivian scheelite, $1.50, domestic scheelite, $12.

Antimony, Beryllium, Iron, Molybdenum, Tantalum, Tin, Titanium, Vanadium, and Zircon ores are unchanged from prices given in the March 23 issue.

Metallic Compounds

ARSENIOUS OXIDE (White arsenic)—Per lb., 4c., delivered, all positions.

COPPER SULPHATE (Blue Vitriol)—Per lb., in car lots, 4.25c. for either large or small crystals.

For prices on Calcium Molybdate, Cobalt Oxide, Germanium Dioxide, Sodium Nitrate, Sodium Sulphate, Zinc Oxide, and Zirconium Dioxide see issue of March 23.

Rolled Metals

COPPER—Per lb., sheets, hot-rolled, 19⅞c.; wire, f.o.b. mill, 11½c.

LEAD SHEETS—Per lb., full rolled, 7½c.; clipped, 7¾c.

MONEL METAL—Per lb., sheets, full finished, 42c.; rods, hot-rolled, 35c.

MUNTZ METAL—Per lb., rods, 16⅜c.; plates, 19¼c.

NICKEL—Per lb., sheets, full finished, 52c.; rods, hot-rolled, 45c.

NICKEL SILVER—Per lb., sheets, 10 per cent, 24⅝@25c.; 18 per cent, 28⅛@28½c. Wire and rods, 10 per cent, 28½c.; 15 per cent, 32⅛c.; 18 per cent, 35⅝c.

PHOSPHOR BRONZE—Per lb., sheets and rods, 5 per cent tin, 31⅛c.; wire, 5 per cent, 31⅝c.; 10 per cent, 37⅛c.

ZINC SHEETS—Per lb., 9½c., f.o.b. works; ribbon, 9c.; 7 per cent discount on order for 18 tons or more.

Metal Statistics

Monthly Average Prices of Metals

Silver

	New York		London Spot		Sterling Exchange	
	1930	1931	1930	1931	1930	1931
January	45 000	29.423	20 896	13.810	486.447	485.260
February	43 193	26.773	20 008	12.432	485.784	485.551
March	41 654	29.192	19.298	13.524	485.933	485.596
April	42 428	.....	19.554	.....	486.019	.....
May	40.736	.....	18 850	.....	485.724	.....
June	34.595	.....	16 049	.....	485.625	.....
July	34 346	.....	15.928	.....	486.300	.....
August	35 192	.....	16.283	.....	486.839	.....
September	36 315	.....	16.738	.....	485.888	.....
October	35.846	.....	16.563	.....	485.719	.....
November	35.908	.....	16.625	.....	485.454	.....
December	32 635	.....	15 201	.....	485.413	.....
Year	38 154	.....	17.666	.....	485.929	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery, Electrolytic		London Spot, Standard		London Spot, Electrolytic	
	1930	1931	1930	1931	1930	1931
January	17.775	9.838	71.469	44.938	83.250	47.524
February	17.775	9.724	71.419	45.372	83.500	47.950
March	17.775	9.854	69.202	44.818	83.405	47.699
April	15 621	.....	62.075	.....	74.338	.....
May	12.756	.....	53.159	.....	59.545	.....
June	12.049	.....	50.003	.....	56.750	.....
July	11.023	.....	48.277	.....	52.522	.....
August	10.693	.....	47.525	.....	50.725	.....
September	10.310	.....	46.264	.....	49.500	.....
October	9.597	.....	43.030	.....	45.772	.....
November	10.113	.....	46.134	.....	48.963	.....
December	10.300	.....	46.771	.....	50.065	.....
Year	12.982	.....	54.611	.....	61.528	.....

New York quotations, cents per pound London, pounds sterling per long ton.

Lead

	New York		St. Louis		London			
	1930	1931	1930	1931	1930 Spot	1930 3 Mos.	1931 Spot	1931 3 Mos.
January	6.250	4.802	6.100	4.604	21.545	21.571	13.872	13.905
February	6.236	4.552	6.086	4.340	21.188	21.097	13.444	13.550
March	5.662	4.527	5.542	4.276	18.807	18.940	13.128	13.349
April	5.537	.....	5.428	.....	18.319	18.363	.....	.....
May	5.523	.....	5.408	.....	17.795	17.861	.....	.....
June	5.410	.....	5.310	.....	17.941	17.994	.....	.....
July	5.250	.....	5.149	.....	18.160	18.063	.....	.....
August	5.488	.....	5.340	.....	18.294	18.178	.....	.....
September	5.500	.....	5.348	.....	17.909	17.798	.....	.....
October	5.151	.....	4.999	.....	15.747	15 674	.....	.....
November	5.100	.....	4.950	.....	15.934	15.931	.....	.....
December	5.100	.....	4.950	.....	15.283	15.292	.....	.....
Year	5.517	.....	5.384	.....	18.077	18.064	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin and Cadmium

	New York		London		Cadmium
	1930	1931	1930	1931	1931
	Straits		Spot		
January	38.851	26.137	175.466	115.798	67.115
February	38.676	26.315	173.750	117.919	55.000
March	36.798	27.065	164.851	121.852	55.000
April	36.077	.....	162.638	.....	.....
May	32.108	.....	144.818	.....	.....
June	30.336	.....	136.300	.....	.....
July	29.822	.....	134.511	.....	.....
August	30.044	.....	134.988	.....	.....
September	29.647	.....	132.631	.....	.....
October	26.802	.....	117.451	.....	.....
November	25.904	.....	113.519	.....	.....
December	25.262	.....	111.560	.....	.....
Year	31.694	.....	141.873	.....	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St Louis		London			
	1930	1931	1930 Spot	1930 3 Mos.	1931 Spot	1931 3 Mos.
January	5.229	4.035	19.634	20.241	12.747	13.113
February	5.180	4.012	19.209	19.778	12.303	12.694
March	4.934	4.002	18.304	18.810	12.190	12.676
April	4.843	.....	17.819	18.378	.....	.....
May	4.641	.....	16 639	17.330	.....	.....
June	4.441	.....	16.422	17.038	.....	.....
July	4.350	.....	16.171	16.777	.....	.....
August	4.360	.....	15.953	16.469	.....	.....
September	4.270	.....	15.773	16.080	.....	.....
October	4.059	.....	14.446	14.935	.....	.....
November	4.266	.....	14.706	15.238	.....	.....
December	4.099	.....	13.762	14.214	.....	.....
Year	4.556	.....	16.570	17.107	.....	.....

St Louis quotations, cents per pound. London pounds sterling per long ton.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1930	1931	1930	1931	1930	1931
January	8.606	7.317	121.192	103.000	61.923	36.000
February	8.830	7.069	120.500	100.205	59.909	34.000
March	8.236	7.127	118.808	100.423	54.769	29.538
April	7.740	.....	114.000	.....	46.462	.....
May	7.454	.....	117.269	.....	46.000	.....
June	7.057	.....	118.600	.....	46.000	.....
July	6.976	.....	117.462	.....	43.692	.....
August	7.839	.....	116.000	.....	41.462	.....
September	7.773	.....	113.620	.....	36.080	.....
October	7.288	.....	110 596	.....	36 000	.....
November	7.105	.....	106.717	.....	36 000	.....
December	7.099	.....	105.346	.....	36.000	.....
Year	7.667	.....	115.009	.....	45.358	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce.

Pig Iron[1] and Aluminum

	Bessemer		Basic		No. 2 Foundry		Aluminum	
	1930	1931	1930	1931	1930	1931	1930	1931
Jan	19.00	17.50	18.50	17.00	18.50	17.00	24.300	23.300
Feb	19.00	17.29	18 50	16.79	18.50	16.79	24.300	23.300
March	19.00	17.00	18.50	16.50	18 50	16.50	24.300	23.300
April	19 00	.....	18.50	.....	18.50	.....	24.300	.....
May	19.00	.....	18.50	.....	18.50	.....	24.300	.....
June	19 00	.....	18 50	.....	18.50	.....	24.140	.....
July	18.62	.....	18 12	.....	18.12	.....	23.300	.....
Aug	18.50	.....	18 00	.....	18 00	.....	23.300	.....
Sept	18.38	.....	17 76	.....	17 88	.....	23.300	.....
Oct	17.85	.....	17 00	.....	17 22	.....	23.300	.....
Nov	17.50	.....	17.00	.....	17.00	.....	23.300	.....
Dec	17 50	.....	17 00	.....	17.00	.....	23.300	.....
Year	18 53	.....	17 99	.....	18.02	.....	23.787	.....

Iron in dollars per long ton. Aluminum in cents per pound, 99 per cent grade

[1]F.o.b. Mahoning and Shenango valley furnaces; freight to Pittsburgh, $1.76.

Monthly Crude Copper Output in Short Tons

Domestic

	1929 Total	1930 Total	1930 Nov.	1930 Dec.	1931 Jan.	1931 Feb.
Alaska shipments	21,947	18,953	1,883	1,817	837	627
Calumet & Arizona	65,246	45,161	3,608	3,166	3,236	3,237
Magma	19,118	15,940	1,940	1,203	1,197	1,202
Miami	29,567	34,568	2,657	2,611	2,348	1,745
Nevada Cons.	133,140	70,990	.....	(d) 16,583	.....	.....
Old Dominion (a)	11,172	10,428	824	879	577	576
Phelps Dodge (b)	111,026	72,308	5,718	5,151	5,155	5,184
United Verde Extens.	29,669	21,908	1,504	1,236	1,412	1,610
Tennessee Corp.	7,870	7,772	667	646	659	585

Foreign

Andes Copper	60,538	47,428	3,749	3,499	.....	.....
Braden	88,155	79,923	9,058	8,588	.....	.....
Boleo, Mexico	12,898	13,940	.....	(d) 3,288	.....	.....
Bwana M'Kubwa	6,988	.....	497	404	.....	.....
Chile	150,247	89,100	7,118	7,121	.....	.....
Furukawa, Japan	17,767	.....	1.432	1,595	1,489	.....
Howe Sound	21,516	22,633	.....	(d) 5,231	.....	.....
Messina	(c) 7,200	.....	826	896	857	.....
Mt. Lyell	9,731	10,878	.....	(d) 2,504	.....	.....
Sumitomo, Japan	20,180	....	1,554	1,367	1,324	.....

(a) Includes Arizona Commercial. (b) Moctezuma is included. (c) Estimated. (d) Three months.

Monthly Production of Primary Copper from U. S. Mines and Daily Rate (Short Tons)

	1929 Monthly Production	1929 Daily Rate	1930 Monthly Production	1930 Daily Rate	1931 Monthly Production	1931 Daily Rate
January	86,324	2,785	67,838	2,188	48,059	1,550
February	84,735	3,026	59,196	2,114	47,504	1,697
March	93,698	3,023	61,216	1,975	.....	.....
April	94,902	3,163	60,450	2,015	.....	.....
May	93,392	3,013	60,238	1,943	.....	.....
June	82,354	2,745	56,743	1,891	.....	.....
July	79,229	2,556	54,249	1,750	.....	.....
August	78,885	2,545	56,779	1,832	.....	.....
September	79,402	2,647	56,584	1,886	.....	.....
October	82,575	2,664	55,954	1,805	.....	.....
November	75,934	2,531	53,141	1,771	.....	.....
December	74,772	2,412	48,518	1,565	.....	.....
Total	1,006,203	.....	690,263	.....	95,563	.....
Monthly average	83,850	.....	57,522	.....	47,782	.....
Av. of daily rate	.....	2,757	.....	1,891	.....	1 620

Mining Share Prices—Month of March, 1931

Company Name	High	Low	Last	§Most Recent Dividend
NEW YORK STOCK EXCHANGE				
Ahumada Lead	0.75	0.50	0.50	April, 1927 0.12½
Alaska Juneau	15	8.12½	14½	Ap. 10, My. 1, 1931 Q 0.10
American Metal	22½	19	20	December, 1930 Q 0.25
Am. Metal, pfd			89*	Fe. 17, Mh. 2, 1931 Q 1.50
Am. Sm. & Ref	56½	44	46½	Ap. 17, My. 31, 1931 Q 1.00
Am. Sm. & Ref. 2nd. pfd.	102¾	100½	100½	My. 8, Je. 1, 1931 Q 1.50
Am. Sm. & Ref., pfd.	138¾	136	137½	My. 8, Je. 1, 1931 Q 1.75
Am. Zinc, L. & S.	7.87½	6.00	6.00	May, 1917 1.00
Am. Zinc, L. & S., pfd.	40¾	35	38	Se. 22, Oc. 1, 1930 Q 1.50½
Anaconda	42¾	32½	33½	..p. 11, My. 18, 1931 Q 0.37½
Andes Copper	18½	14¼	14¾	Ja. 10, Fe. 9, 1931 Q 0.25
Butte C. & Z.	1.50	1.00	1.00	De. 10, De. 25, 1929 0.50
Butte & Superior	2.00	1.75	1.75	De. 13, De. 31, 1929 Q 0.50
Callahan Zinc-Lead	1.37½	1.00	1.00	December, 1920 0.50
Cal. & Arizona	43¾	38½	38½	June, 1930 0.50
Cal. & Hecla	11	9.00	9.00	June, 1930 0.50
Cerro de Pasco	28½	23½	24	Ja. 15, Fe. 2, 1931 Q 0.50
Dome Mines	13½	10¾	12¾	De. 31, Ja. 20, 1931 Q 0.25
Federal M. & S.	75	75	75	June, 1927 10.00
Federal M. & S., pfd			94*	Fe. 26, Mh. 16, 1931 Q 1.75
Freeport Texas	43½	35½	37½	My. 15, Je. 1, 1931 Q 0.75
Granby Con	22	17½	17½	Ja. 16, Fe. 2, 1931 Q 0.50
Greene Cananea	74¾	74¾	74¾	July, 1930 0.75
Homestake	104	85	101	Mh. 20, Mh. 25, 1931 M 0.50
Howe Sound	29	24¾	24¾	Mh. 31, Ap. 15, 1931 Q 0.75
Inspiration	11	8.87½	8.87½	July, 1930 0.50
International Nickel	20¼	17¼	18	Mh. 2, Mh. 31, 1931 Q 0.15
Int. Nickel, pfd	123	120	123	Ja. 3, Fe. 2, 1931 Q 1.75
Kennecott	31½	24¾	24¾	Mh. 12, Ap. 1, 1931 Q 0.50
McIntyre-Porcupine	26½	21	26	Fe. 2, Mh. 2, 1931 Q 0.25
Magma Copper	26½	20¾	20¾	Mh. 31, Ap. 15, 1931 Q 0.50
Miami Copper	10½	8.00	8.00	August, 1930 0.375
Mother Lode	0.75	0.50	0.50	June, 1930 0.10
National Lead	128	120	120½	Mh. 13, Mh. 31, 1931 Q 1.25
National Lead, pfd. A	141	140	140½	Fe. 27, Mh. 14, 1931 Q 1.75
National Lead, pfd. B	119¾	118	119¾	Ap. 17, My. 1, 1931 Q 1.50
Nevada Con	14¾	11¼	11¾	Mh. 13, Mh. 31, 1931 Q 0.25
Park Utah Con	2.25	1.75	1.75	Je. 14, Jl. 10, 1929 Q 0.20
Patiño M. & E.	14	11½	11½	December, 1929 4s.
Phelps Dodge	24½	21½	21½	De. 13, Ja. 2, 1931 Q 0.50
Rand Mines	27½	27½	27½	August, 1930 1.52
St. Joseph Lead	29	22½	22½	Mh. 10, Mh. 20, 1931 Q 0.50
Seneca Copper	1.75	1.25	1.25	
Tennessee Corp	9.12½	7.50	7.50	Fe. 28, Mh. 16, 1931 Q 0.125
Texas Gulf Sulphur	54	49¼	49¾	Mh. 2 Mh. 16, 1931 Q 1.00
U. S. S. R. & M.	25¾	20	21¼	Ap. 2, Ap. 15, 1931 Q 0.25
U. S. S. R. & M., pfd.	46¾	44¾	46¾	Ap. 2, Ap. 15, 1931 Q 0.875
Utah Copper	122	115	122	Mh. 13, Mh. 31, 1931 Q 2.00
Vanadium Corp	76¼	62¾	66¼	Fe. 2, Fe. 16, 1931 Q 0.75
NEW YORK CURB EXCHANGE				
Aluminum Co. of Am.	224	155	191	
Alum. Co. of Am., pfd.	109¾	109	109½	De. 15, Ja. 1, 1931 Q 1.50
Anglo-Chilean Con	15	10½	11	
Bunker H. & S.	50	49½	49½	Mh. 26, Ap. 16, 1931 M 0.25
Carnegie Metals	2.00	2.00	2.00	
Con. Coppermines	3.25	2.87½	3.12½	
Copper Range			7.75*	July, 1930 0.25
Cresson Con	0.62½	0.37½	0.50	Mh. 31, Ap. 10, 1931 Q 0.01
Cusi Mexicana	1.12½	0.50	0.87½	
Evans-Wallower	2.87½	1.50	1.50	
Federated Metals	10½	10½	10½	Ja. 5, Ja. 15, 1931 Q 0.25
Hecla Mining	7.00	6.62½	7.00	Fe. 15, Mh. 15, 1931 Q 0.10
Hollinger Con	8.62½	7.62½	8.62½	My. 6, My. 20, 1931 M 0.05
Hudson Bay M. & S.	6.37½	5.12½	5.75	
Iron Cap Copper	1.75	0.87½	1.50	
Mayflower Associates	50	43½	48	Mh. 2, Mh. 16, 1931 Q 0.50
Mining Corporation	2.87½	2.50	2.50	De. 4, De. 19, 1929 0.12½
New Jersey Zinc	50	46¾	48	Ap. 20, My. 9, 1931 Q 0.50
Newmont Mining	57	46	47	Mh. 31, Ap. 15, 1931 Q 1.00
N. Y. and Honduras	12	11	12	Ap. 14, Ap. 25, 1931 QX 0.50
Nipissing	1.87½	1.50	1.62½	Mh. 31, Ap. 20, 1931 Q 0.075
Noranda Mines	29¾	17	24½	October, 1930 0.50
Ohio Copper	0.50	0.37½	0.37½	September, 1926 0.03
Premier Gold	0.75	0.68¾	0.75	Mh. 13, Ap. 4, 1931 Q 0.03
Quincy			6.00*	March, 1920 1.00
Shattuck Denn	6.00	4.87½	4.87½	
So. Am. Gold & Plat.	1.25	1.12½	1.12½	
Teck-Hughes	8.25	6.87½	8.25	Ja. 17, Fe. 1, 1931 Q 0.15
Tonopah Mining	0.43¾	0.37½	0.43¾	Se. 30, Oc. 21, 1929 0.07½
United Eastern			0.06¼*	
United Verde Ex	13¾	11¾	13¾	Ap. 2, My. 1, 1931 Q 0.50
Utah-Apex	1.12½	1.12½	1.12½	De. 15, No. 1, 1929 0.025
Yukon Gold			0.12½*	June, 1918 0.02
BOSTON STOCK EXCHANGE				
Ariz. Commercial	0.85	0.70	0.75	Fe. 21, Fe. 28, 1931 IL 1.00
East Butte	0.40	0.15	0.25	Fe. 16, Mh. 4, 1931 L 2.00
Isle Royale	6.00	4.50	4.50	March, 1930 0.50
Mohawk Mining	21	17	19½	Ja. 31, Fe. 28, 1931 Q 0.25
Old Dominion	3.75	3.00	3.00	December, 1918 1.00
Utah Met. & Tunnel	0.57	0.35	0.40	December, 1927 0.30
MONTREAL				
Asbestos Corp	0.50	0.40	0.50	January, 1926 1.50
Asbestos Corp., pfd	2.00	1.25	1.25	January, 1930 1.75
Consolidated M. & S.	188¼	163	163	De. 30, Ja. 15, 1931 SAX 6.25

Company Name	High	Low	Last	§Most Recent Dividend
SPOKANE				
Constitution	0.07	0.05	0.06	
Dayrock	1.30	1.05	1.05	March, 1930 0.03
Golconda	0.45	0.36½	0.38	July, 1929 0.02
Jack Waite Con	0.43½	0.39½	0.40	
Los Lugos	0.03½	0.01½	0.01⅛	
Mexican Premier	0.19	0.14	0.14	January, 1930 0.01
Montana Mines	0.16	0.12	0.15½	
Noble Five	0.07	0.05	0.05½	January, 1930 0.005
Pend Orielle	1.50	1.10	1.20	
Sherman Lead	0.45½	0.39	0.39	March, 1930 0.01
Sidney	0.12½	0.07	0.09	October, 1929 0.01
Sunshine Mining	1.61	1.15	1.45	December 20, 1930 Q 0.03
Tamarack & Custer	0.29	0.20	0.22	September, 1924 0.25
SALT LAKE CITY				
Chief Con	0.72	0.60	0.63	February, 1928 0.10
Combined Metals	0.10	0.09½	0.09½	
Eureka Bullion	0.19½	0.15½	0.16	
Eureka Lilly	0.19½	0.15½	0.15½	
Moscow Silver	0.45	0.35	0.35	
New Quincy	0.05½	0.02½	0.02½	Se. 20, Se. 30, 1929 Q 0.10
North Lily	1.10	0.95	0.95	January, 1930 0.15
Park Bingham	1.25	1.25	1.25	
Park City Con	0.25½	0.21	0.21	
Rico Argentine	0.04	0.03¾	0.04	December, 1929 0.03
Silver King Coal	5.40	5.00	5.05	De. 20, Ja. 2, 1931 Q 0.15
Tintic Lead	1.40	1.15	1.35	
Tintic Standard	4.30	3.95	4.10	March 31, 1931 Q 0.10
Walker Mining	2.00	1.75	1.75	April, 1930 0.075
TORONTO				
Abana Mines	0.21	0.13½	0.16½	
Amulet Mines	0.45	0.36	0.38	
Base Metals	0.19½	0.14	0.15	
Barry-Hollinger	1.35	1.15	1.15	
Canam Metals	0.05	0.03½	0.03½	
Castle-Trethewey	0.26½	0.18	0.21	
Central Manitoba	0.23	0.08	0.15	
Falconbridge	1.69	1.45	1.60	
Howey Mines	0.33	0.28	0.29	
Keeley	0.34	0.25	0.25	March, 1928 0.04
Kirkland Lake	0.78	0.57	0.77	
Lake Shore	28½	25	28½	Mh. 2, Mh. 16, 1931 Q 0.30
Sherritt-Gordon	1.20	1.04	1.10	
Siscoe Gold	0.53	0.43	0.45	
Sylvanite	1.07	0.56	1.06	De. 5, De. 20, 1930 SA 0.02
Treadwell-Yukon	2.25	2.20	2.25	
Ventures	0.86	0.52	0.70	
Wipond Con	1.19	1.05	1.11	April, 1927 0.03
Waite A.-M	1.75	1.50	1.70	
Wright-Hargreaves	2.70	1.98	2.62	Mh. 14, Ap. 1, 1931 0.025
ST. LOUIS, CINCINNATI, AND DENVER				
Consolidated L. & Z.	3.00	2.00	2.62½	December, 1929 0.125
Eagle-Picher	6.25	5.00	6.00	April, 1930 0.20
Eagle-Picher, pfd			100*	De. 31, Ja. 15, 1931 Q 1.50
Empire Chief	0.02	0.01½	0.01½	
Star of Utah	0.58	0.40	0.40	
Tungsten Prod	0.20	0.09	0.09½	
West Gold	0.03	0.02	0.03	
SAN FRANCISCO, LOS ANGELES, AND COLORADO SPRINGS				
California Copper	0.75	0.50	0.50	
(a) Golden Cycle	14½	14½	14½	Fe. 28, Mh. 10, 1931 Q 0.40
(a) Granite Gold	0.20	0.19	0.19	March, 1931 Q 0.01
Idaho-Maryland	0.62½	0.33½	0.42½	
Tom Reed	1.27½	0.87	1.27½	Mh. 16, Ap. 1, 1931 0.03
(a) United Gold	0.08½	0.08¼	0.08½	February, 1930 0.01

‡LONDON

Company Name	High	Low	Last	Most Recent Dividend (Per Cent, unless noted)
Anglo American	18/1½	14/6	18/1½	22½, Feb. 19, 1930
Aramayo Mines	32/8	21/3	27/6	5, Oct. 23, 1929
Burma Corporation	10/1	8/4½	9/4½	1¼, Dec. 10, 1930(b)
Bwana M'Kubwa	6/6	5/10½	6/3	
Camp Bird	7d	5d	6d	16¼, April 25, 1928
(a) Fresnillo	8/9	7/6	8/3	25 cents, Feb. 12, 1930
Frontino & Bolivia	10/6	8/9	8/9	3¾, January, 1929
Lake View & Star	11/–	9/–	10/6	11¼, 1921
Mexican Corporation	5/6	3/10½	5/–	10, Oct. 24, 1929
Oroville Dredging	1/10½	1/9	1/9	37½, May 16, 1930
Rhodes. Congo Border	156/6	135/–	155/–	
Rhodes.-Select. Trust	20/–	16/9	20/–	
Rio Tinto	628/9	550/–	572/–6	10, Nov. 21, 1930
St. John del Rey	20/9	19/3	20/6	3¾, Nov. 21, 1930
San Francisco Mines	14/4½	12/6	13/3	10, Jan. 7, 1931
Santa Gertrudis	8/3	6/9	8/–	7½, Jan. 8, 1931
Selukwe	3/–	2/7½	3/–	6¼, April 2, 1917
So. American Copper	3/2¼	2/7½	2/7½	
Tanganyika	36/—	29/4½	31/10½	2½, Aug. 2, 1930
Wiluna Gold	18/10½	15/3	18/6	
Union Minière	6,675	6,105	6,115	300 francs, July, 1930

Salt Lake Stock Exchange courtesy J. A. Hogle & Company, Spokane Standard Exchange courtesy the Pohlman Investment Company. *Not traded in. †Belgian Francs. ‡The above London quotations are for the month ended Mar. 20. Q, quarterly; I, initial; M, monthly; X, extra; SA, semi-annual; L, liquidating. §Total dividends paid during the month will be found tabulated in the Market and Financial News Section. Dates here given are those of closing of books and payment of dividend. (a) Bid prices. (b) 1¼% (2 annas) dividend, and bonus of ¾% (1 anna).